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TO A NEW MOVEMENT, A BROAD ROAD!

The noble initiative of the Hero of Socialist Labor Valentin Gaganova has, after a short time, inspired succeeding Soviet metallurgists. Many very experienced foremen and team-leaders, regardless of difficulties, have transferred to lagging teams and shifts and exerted every effort to make them leaders.

The labor record of the distinguished steel-smelter of the "Dneprospetsstal'" factory, Dmitrii Galushko, is well known to the Ukraine. But this experienced steel founder has not rested on his achievement. For how can one remain calm if a collective of neighboring teams systematically does not fulfil the plan and tolerates great waste? So, Dmitrii Galushko took over a furnace team which was among the laggards. His abundant experience in production and ability to organize people enabled him to lead the furnace collective out of its slump. This team not only fulfilled the plan in August, but produced about 200 tons of high-grade steel over its quota.

An experienced team-leader of the same factory, Fedor Chistyakov, took over the leadership of another collective of steel founders which had been lagging. In August his team produced more than 70 tons of high-grade steel above the plan.

At the "Zaporozhstal'" the initiative of V.Gaganova first inspired the well-known blast-furnace foreman, M.P. Voloshin, the team-leader of the picklers in the cold-rolling shop, M. Seregin, and others.

The example of Valentin Gaganova was followed by Dnepropetrovsk metallurgists: Alexei Shpak, foreman of the open-hearth plant of the Lenin Nizhnedneprovsk factory, Leonid Kushnirskii, the foreman of the open-hearth plant of the Petrovskii factory, Nikolai Velikii, foreman of shop No. 1 of the Nikopol' southern pipe factory, and others.

Followers of this extraordinary patriot also appeared in the Kirov factory at Makeevka. Petr Gorbunov, agglomerate foreman Leonid Slez, shift chief of No. 1 open-hearth, Grigorii Brevko, senior operator of rolling mill No. 2, and others transferred from leading collectives to work in lagging ones.

The workers in previously lagging teams welcomed their new leaders; they did all they could to make the experienced foreman acquainted with the new place and helped him to overcome the arrears.

With the arrival of the experienced leaders, the collectives of the teams began to urgently seek and use productive reserves, to demonstrate creative initiative and activity. Their confidence in their powers, and in the struggle which had been begun rose each day.

"Following your noble initiative," wrote Anatolii Serkov, Hero of Socialist Labor and steel founder at the "Red October" factory, to Valentin Gaganova, I transferred from the team of a leading Martin furnace to a lagging one. Consulting with the members of my team, we decided to make the furnace one of the leaders, to fulfil the 1959 production plan ahead of time on all points: to smelt 1000 tons of high-grade steel by the end of the year. At the same time we challenge the team headed by you to socialist competition."

Viktor Golubitskii, foreman of open-hearth plant No. 3 of the Dzerzhinskii factory, transferred from a leading team of the casting area to a lagging one. In a short time he strengthened the working and technological discipline in the team and achieved an improvement in the production indices. With each day the collective is speeding up its pace, obtaining a rhythmic operation of the furnace and confidently becoming one of the leaders.

A.F. Grebenyuk, a senior rolling-mill operator at the Stalin Metallurgical Factory was one of the first at that enterprise to transfer to a lagging team, following the example of V. Gaganova. Even on the first day of work A.F. Grebenyuk noticed that mutual assistance was lacking, that efforts were not properly distributed, that there were no replacements for especially heavy sections as was done in other teams. The senior operator, after consulting with the foreman, decided to conduct a meeting of the team. This first meeting went very well. The workers listened carefully to the remarks of the new leader and agreed with all his suggestions. On the spot they set up and approved a substitution schedule and also made some changes in the arrangement of working places. A.F. Grebenyuk is urgently teaching the workers to save time and is transmitting his abundant experience to them. Tactful and attentive to people, demanding toward himself and his comrades, the senior operator has reached the spirit of the workers, and although not everything is running smoothly yet in the team, it is already on the right path: the July plan of the collectives was fulfilled by 100.4% and the day is not distant when it will become a leading team.

Mikhail Zheinerovich, team-leader of the calibrators, has been working more than fourteen years at the Serov Metallurgical Combine. During this time he joined a number of the leaders who have an excellent knowledge of the business of production. His team was one of the first in the combine to enter the struggle to be properly called collective communist labor.

Not long ago, along with other followers of Valentin Gaganova, he transferred to a lagging team. M. Zheinerovich began with the application of discipline, the organization of educational work and the distribution of leading experience. In the first week the new team-leader called to order some who liked to come to work a little late, to sit in the dining room a little longer and to go home a little early. Discussions began to be held regularly in the team, which had not been done previously. M. Zheinerovich himself led the first discussion, on "Friendship, Comradeship and Discipline."

Shortly afterward, at one of the team discussions, he introduced a series of valuable suggestions directed at shortening the time in rebuilding machinery and preparing metals for wire-drawing. In June the previously lagging team paid off its arrears, 28 tons of calibrated steel, and successfully fulfilled the state plan.

The experience of the followers of Valentin Gananova once more proves quite conclusively that there are no poor workers and, even more so, there are no poor collectives. The success of any affair depends on ability to organize people, mobilize them for the fulfillment of a task and bring exemplary order into enterprises.

If a capable manager heads an enterprise, one who shows himself to be a personal example of high communist consciousness, one can be confident that success will be attained.

A patriotic movement has presented the trade union organizations of metallurgists with a whole series of new tasks. People who have decided to assist their comrades and take in hand lagging sections of production are encountering a great deal of difficulty, to work under new conditions, with new people, is not an easy matter. The party and trade union organizations must help them in every possible way.

The remarkable example of Gaganova can not only lead to many individual leading sections and members of plants in metallurgical factories but even put an end to lagging of entire enterprises and branches of the ferrous metal industry. This will be an enormous contribution to the matter of building communism in our country.

THE BLOWING OF THE FIRST BLAST FURNACE AT THE BHILAI STEEL PLANT IN INDIA

G.F. Mikhalevich and I.I. Sagaidak

On February 3, 1959, the No. 1 blast furnace of useful volume 1033 m^3 of the Indian State Steel Plant in Bhilai was put into operation. This plant was built in accordance with the standard design of Gipromez. The plans included operation of the blast furnaces using fluxed sinter.

Raw materials. The raw materials can by unloaded into the trench of the oreyard from an end-mounted elevated rail or directly into the bunkers. After grading, the coke is fed to the elevated rail by a 1200 mm wide conveyer and is unloaded into the bunkers. The ore bunkers have drum type gates and the coke bunkers have vibration grates.

The plant uses ores from the Dhalli Rajhara deposits which are about 100 km from the plant. Four types of ores are mined here: 1) strong solid hematite; 2) stratified hematite; 3) lateritic; 4) nodular.

The bulk of the ores belong to the first two groups and their characteristics are given in Table 1.

At the present time nodular ores are used for the smelting and they are ground manually to a size of 80 mm; the material which is less than 15 mm is screened off, also manually.

The limestone deposits are 24 km from the plant.

The limestone compositions show a wide variation in the content of components and increased content of SiO_2 and Al_2O_3. In places they are dolomitized.

The average composition of the limestone is given below, %:

CaO	SiO_2	Al_2O_3	MgO	Fe_2O_3	P	S
42	7	2	8	1	0.013	Traces

The limestone for smelting is also ground manually.

The construction of grinding and grading plants at the ore plants and the open-cut mines will be completed in 1960. The quartzite which is mined and ground manually 72 km from the plant has the following composition, %: 97.65 SiO_2; 0.85 Fe_2O_3; 0.24 Al_2O_3.

The manganese ores are supplied from ore plants which are 250 km from the plant. The average analysis of the manganese ores is as follows, %:

Mn	Fe	SiO_2	Al_2O_3	CaO
35	6.4	20.6	1.8	1.10

The charge for coking uses 60% of the Jharia coals and 40% of the Bokaro coals. The Jharia and Bokaro deposits (in the province of Bihar) are 720-750 km from the plant.

The coal compositions are as follows, %:

	Moisture	Ash	Sulfur	Phosphorus	Volatiles
Jharia	1–2	17–20	0.4–0.6	0.1–0.3	–
Bokaro	2–3	18–25	0.5–0.8	0.25–0.3	24–28

Washing the coals of the Bokaro deposits reduces the ash content by an average of 3%.

Coke of the following composition is obtained from the coal, %:

Ash	Sulfur	Volatiles	Moisture
23.5–26.0	0.6	1.5–2.0	4–6

The drum test was 320-325 kg.

The coal loader at the coal yard is not yet working, which means that the charge for coking is only partially mixed, i.e., insufficiently.

Blowing the furnace. After the equipment had been tested for pressure, the blast furnace was dried. Heating spirals of iron wire of total power 75 kw were placed over the hearth. The drying lasted 72 hours.

TABLE 1. Chemical Composition of Ores

Type of ore	Content, %				
	Fe	SiO_2	Al_2O_3	P	S
Solid	68.2	0.71	0.77	0.064	Traces
Stratified	62.4	2.83	2.51	0.068	"
Lateritic	48.5	8.20	10.14	0.175	"
Nodular	66.2	2.08	2.03	0.05	"

The crowns were heated up to 500° by means of air heaters using firewood and then up to a temperature of 1100° using oil. The consumption of firewood was 250 m^3 and that of oil 340 tons. On February 2nd, after preliminary cooling of the furnace, the hearth was filled with dry firewood up to the level of the tuyeres and the furnace was then charged. The start-up charges were estimated at twice the volume of the furnace, with gradual increase in the ore load (Table 2).

TABLE 2. Calculated Composition of Blow Charge

No. of charge	Quantity of materials, tons					Ore load	Yield of iron from charge	Yield of slag from charge
	coke	iron ore	manganese ore	limestone	quartzite			
1	75.0	—	—	—	—	0.00	—	16.3
2	75.0	—	—	53.0	—	0.00	—	46.8
3	75.0	30.0	1.0	39.0	6.0	0.40	23.9	43.1
4	75.0	35.0	1.1	39.0	6.4	0.47	27.6	43.4
5	75.0	40.0	1.3	40.0	7.0	0.53	31.2	44.6
6	75.0	45.0	1.5	40.0	7.2	0.60	34.9	44.8
7	75.0	50.0	1.6	41.0	7.3	0.67	38.5	45.5
8	75.0	55.0	1.8	41.0	7.8	0.73	42.1	45.1
9	75.0	60.0	1.9	42.0	7.5	0.80	45.8	46.6
10	75.0	65.0	2.0	42.0	7.5	0.87	49.2	46.8
11	75.0	70.0	2.1	43.0	7.5	0.93	52.9	47.6
12	75.0	75.0	2.3	43.0	7.5	1.00	56.5	47.9
1—6	450.0	150.0	4.9	211.0	26.6	0.33	117.6	239.0
7—12	450.0	375.0	11.6	252.0	44.6	0.83	285.0	280.1
1—12	900.0	525.0	16.5	463.0	71.2	0.58	402.6	519.1

The following data were used to calculate the blow charges. The composition of the raw materials (from samples taken at the bunker gates) and ash of the coke, %:

	Fe	Mn	SiO_2	Al_2O_3	CaO	MgO
Iron ore	67.20	—	1.96	1.32	—	—
Manganese ore	7.70	33.94	21.23	6.90	—	—
Limestone . . .	1.20	—	5.62	1.90	43.70	6.26
Quartzite	2.10	—	93.7	2.19	2.00	—
Coke ash . . .	5.00	—	54.7	31.00	5.20	—

Content in the coke, %: ash (from samples taken from the coke conveyer) 24.00 and sulfur 0.6.

Degree of conversion of iron to pig iron -- 99.5%.

Degree of conversion of manganese into the pig iron—70.0%. Basicity of slag $CaO:SiO_2$—1.1.

Content of alumina in the slag—16.0%.

Degree of conversion of sulfur into the slag—82%. Content of Si in the iron—3—4.5%, Mn—1.0%.

Table 3 shows the actual charge which was loaded into the blast furnace during start up and in the first period of operation.

TABLE 3. Consumption of Raw Materials and Coke, tons and Charging System

No. of feed	Coke	Iron ore	Manganese ore	Quartzite	Limestone	Charging system
1	7.5	—	—	—	—	CCC↓
11	7.5	—	—	—	5.3	CCCC↓
21	7.5	3.0	0.50/5*	1.20/2**	3.9	OCCC↓
31	7.5	3.5	0.55/5	1.30/2	3.9	OCCC↓
41	7.5	4.0	0.65/5	1.40/2	4.0	OCCC↓
51	7.5	4.5	0.75/5	1.45/2	4.0	OCCC↓
61	7.5	5.0	0.80/5	1.45/2	4.1	OLCCC↓
71	7.5	5.5	0.90/5	1.45/2	4.1	COLCC↓
81	7.5	6.0	0.90/5	1.50/2	4.2	COLCC↓
89	7.5	6.0	0.90/5	1.50/2	3.9	COLCC↓
91	7.5	6.5	1.00/5	1.50/2	3.9	COLCC↓
98	7.5	6.5	1.00/5	1.50/2	3.7	COLCC↓
101	7.5	7.0	1.10/5	1.50/2	3.8	COLCC↓
111	7.5	7.5	1.15/5	1.50/2	4.0	COLCC↓
120	7.5	7.5	1.15/5	1.50/2	4.2	COLCC↓
131	7.5	8.5	1.20/5	1.50/2	4.2	COLCC↓
141	7.5	8.5	1.20/5	1.50/2	4.3	COLCC↓
148	7.5	8.5	1.20/5	1.40/2	4.6	OLCC↓
162	7.5	8.3	1.20/5	1.40/2	4.6	OLCCC↓
192	7.5	8.5	1.20/5	1.40/2	4.6	OLCCC↓
210	7.5	8.5	1.20/5	1.40/2	4.6	COLCC↓
218	7.5	8.5	1.20/5	1.40/2	4.6	OLCCC↓
245	7.5	9.0	1.20/5	1.40/2	4.2	OLCCC↓
272	7.5	9.0	1.20/5	1.40/2	4.4	OLCCC↓

*0.50 tons in each fifth feed.

**1.20 tons in each second charge.

The furnace was charged with 62 feeds: 465 tons of coke, 160 tons of iron ore, 5.2 tons of manganese ore, 28.1 tons of quartzite and 219 tons of limestone. In the last feed, which was loaded according to the system CCC↓ O↓, the profile of the charge was measured at the throat.

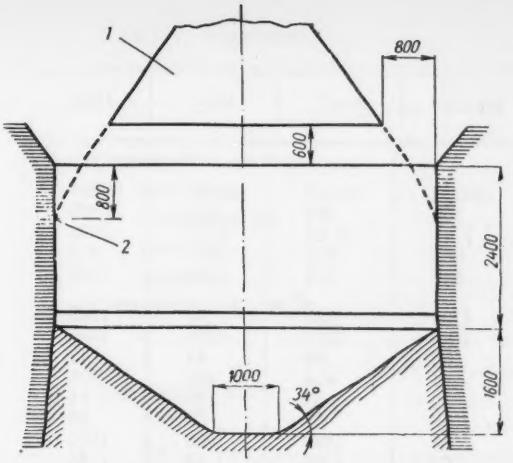
The results of the measurements (diagrams) show that the charge is distributed evenly along the periphery of the furnace with the formation of a funnel which is about 1600 mm deep.

The ore, limestone and quartzite are distributed uniformly across the section of the throat. An area of diameter 1 m in the center of the furnace contains no ore and is filled with coke.

At 0 hr 55 min on February 3rd, 1959, air heated to 800° was blown into the furnace at the rate of 1600 nm^3/min. After the firewood had burnt, the blast was reduced to 600 nm^3/min.

At 7 hr, slag appeared from the iron tapping hole into which an iron pipe had been placed. At 8 hr 15 min the tapping hole was closed. Tables 4 and 5 give data on the operation of the furnace after it had been blown.

It can be seen from these results that the furnace had taken the blast; the feeds were running together uniformly without any holdups.



Profile of charge at the throat before start up: 1) large bell in the lowered position; 2) the point where the trajectories of the poured raw materials meet the segments of the throat.

The compositions of the iron and slag were normal and differed very little from the calculated values.

From February to June 1959 the furnace melted cast iron containing between 1.5 and 3.5% silicon, depending on the orders.

Table 6 gives the average monthly indices for the furnace operation.

The reduction in output for June was due to changeover of smelting of higher silicon content irons.

The index for the direct reduction R'd was 46.2%. The insufficient development of the indirect reduction reactions can be explained by the fact that dense and difficultly reducible closely ground ore was used.

The insufficient development of the indirect reduction reactions and the high ash content in the coke (24-26%) led to an increased specific consumption of coke.

The coke consumption also increased because of the high consumption of limestone due to the introduction of quartzite into the charge. Quartzite was added to

TABLE 4. Indices of Control and Measuring Instruments

Data	3rd February								4th February			5th February			6th February
Time	1	2	3	4	5	6	7—14	15—22	23—6	7—14	15—22	23—6	7—14	15—22	23—6
Pouring level, m	3.0	*	*	*	*	2.5	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75
Number of feeds**	0	3	7	10	13	16	38	57	85	105	126	145	163	185	215
Pressure of blast, atm	0.75	0.40	0.30	0.35	0.37	0.30	0.30	0.38	0.41	0.60	0.70	0.65	0.65	0.70	0.60
Temperature of blast, °C	800	555	545	595	590	600	550	545	575	630	650	650	655	650	645
Temperature of throat, °C	35	35	37	37	43	37	40	140	190	255	280	235	220	260	260

*No data.

**After blowing.

TABLE 5. Characteristics of Iron and Slag

No. of iron tapping	Date	Time hours-min	No. of ladles with iron*	Composition, %						Basicity of slag $\frac{CaO}{SiO_2}$	Remarks
				iron			slag				
				Si	Mn	S	SiO_2	Al_2O_3	CaO		
—	3/II	8—00	—	—	—	—	32.0	15.8	43.0	1.34	From iron tapping hole
—	3/II	16—30—18—00	—	—	—	—	35.7	16.0	41.6	1.17	From slag tapping hole 1
1	3/II	21—00—21—25	0.75	4.0	1.47	—	40.0	13.9	40.0	1.00	
2	4/II	2—45—3—20	0.75	3.5	1.19	—	36.9	14.9	41.8	1.13	
3	4/II	8—55—9—55	1	4.3	0.87	0.041	37.4	15.4	43.2	1.15	
4	4/II	15—30—16—25	1.25	4.8	0.94	0.043	40.1	18.3	36.4	0.91	
5	4/II	23—00—23—45	1.5	4.0	0.66	0.042	40.3	17.4	37.4	0.93	
6	5/II	8—25—9—25	2.25	3.5	1.03	0.049	38.7	15.9	33.0	0.88	
7	5/II	14—45—15—40	1.5	3.4	1.26	0.056	36.6	15.4	39.5	1.08	
8	5/II	23—15—0—10	2.25	2.9	1.25	0.054	39.6	15.5	36.5	0.92	
9	6/II	7—55—9—40	2.75	2.6	0.91	0.063	37.4	16.4	38.0	1.02	

*The scales for the hot iron were not yet in operation

TABLE 6. Indices for Furnace Operation

Indices	February	March	April	May	June
Iron smelted, tons:					
for a month	16700	24220	27158	30531	28854
average for a day	622	782	905	985	962
coefficient of utilization of useful volume	1.44	1.15	0.99	0.91	0.93
Yield of slag, kg/ton iron	685	650	620	600	609
Intensity of coke combustion, kg/m^3. day	583	837	945	1110	1085
Consumption, kg/ton:					
coke	970	1105	1080	1123	1165
iron ore	1305	1455	1372	1390	1465
manganese ore	41	60	56	44	43
limestone	563	567	494	495	505
quartzite	57	29	39	41	41
Dust removal, kg/ton iron	16	36	51	52	29
Consumption of blast, m^3/min	1280	1635	1850	1990	1980
Pressure of blast, atm	0.72	0.93	1.03	1.13	1.43
Temperature of blast, °C	700	750	750	750	770
Humidity of blast, g/m^3	No data				19.6
Gas pressure at throat, atm	0.10	0.18	0.23	0.31	0.64
Temperature of gas at throat, °C	230	280	300	340	345
Composition of iron, %					
Si	2.76	2.03	2.18	1.87	2.16
Mn	0.98	1.16	1.10	0.95	0.97
S	0.034	0.027	0.025	0.031	0.029
P	0.17	0.15	0.18	0.18	0.18
Composition of slag, %:					
SiO_2	35.8	34.4	34.4	35.5	34.8
Al_2O_3	17.1	18.6	19.2	18.9	19.2
CaO	37.0	37.0	36.7	35.4	36.0
MgO	6.8	6.5	6.3	6.8	6.2
Basicity of slag $\frac{CaO}{SiO_2}$	1.03	1.08	1.07	1.00	1.03
Idle time of furnace, hr-min	16—45	35—40	39—40	14—55	11—00
Slow running, hr-min*	590—00	483—00	158—00	2—25	18—15
Changing air tuyeres	—	2	1	—	—
Changing slag tuyeres	2	2	3	2	4

*Consumption of blast less than 1800 m^3/min

TABLE 7. Indices of Furnace Operation with Low and Increased Gas Pressure

Indices	Low pressure	High pressure
Time of operation, days	29	29
Average daily output, tons	993	996
Actual corrected output*	1015	1030
Coke consumption, kg/ton	1130	1130
Slag yield, kg/ton	593	609
Dust removal, kg/ton	53	32
Blast consumption, m^3/min	1995	1995
Blast pressure, atm	1.10	1.43
Temperature, °C:		
blast	750	768
throat	336	343
Gas pressure at throat, atm	0.26	0.61
Drop in pressure, atm	0.84	0.82

* Corrected output of furnace taking into account shutdowns and slow running: $\frac{993 \cdot 1440}{(1440-31.5)}$ = 1015 ton; $\frac{996 \cdot 1440}{(1440-45.7)}$ = 1030 ton. "Slow running" is taken as operation at half speed.

TABLE 7. (continued)

Indices	Low pressure	High pressure
Consumption, kg/ton:		
iron ore	1378	1465
manganese ore	45	38
limestone	495	510
quartzite	40	39
Composition of iron, %:		
Si	1.88	2.04
S	0.028	0.034
Composition of slag, %		
SiO_2	35.3	35.1
CaO	35.5	35.7
Basicity of slag CaO : SiO_2	1.00	1.02
Furnace shutdown, min/day**	27.2	27.2
Slow running, min/day	8.7	37.0

** Pressure of blast less than 0.3 atm.

prevent the alumina content in the slag from exceeding 20%, which ensured an increase in mobility of the slag.

When construction work on the sintering plant is finished, the specific consumption of coke will definitely be reduced.

The large number of shutdowns and in particular the slow running in February and March were due to the shortage of coke and also to the fact that the ladle station and the second casting installation had not been finished.

Furthermore, the plant did not have lime of the required quality, a fact which introduced difficulties into the operation of the casting installation due to extensive cracking of the steel molds.

After some of these adverse conditions had been removed, the time of shutdowns and the slow running were considerably reduced; the indices of furnace operation were improved.

After May 26th, the furnace began operating with increased gas pressure (0.61 atm). This meant that the amount of throat dust removed could be reduced by 40% with unchanged coke consumption, despite a certain reduction in iron content in the charge (Table 7).

On the basis of the start up and the first months of operation at the Bhilai blast furnace, the following conclusions can be drawn.

1. The starting up was normal.

2. Further improvements in the furnace operation can be achieved by reducing the ash content in the coke, by improving the quality of the coals used for coking, by washing the coals, and also by using low ash coals of the Korba deposits, as is provided for in the plan. It is also essential to start operation with fluxed sinter, as is also provided for in the plan. The iron ore and limestone should be crushed about twice as fine as is the case at the present time. This will be possible after the commencement of crushing and grading plants.

As a matter of interest, sharp dissatisfaction has been expressed in the Indian Press with regard to the poor operation of the No. 1 blast furnace in Rourkela, which has been built by a West German firm. The Rourkela furnace is equivalent in capacity to the Bhilai furnace but during the 5 months of operation it has smelted half the quantity of iron smelted by the Bhilai furnace. In addition, the quality of Rourkela furnace production is much poorer than that of the Bhilai furnace.

COST ACCOUNTING RELATIONSHIPS BETWEEN A SINTERING PLANT AND A BLAST FURNACE SHOP

A.D. Kissin and D.A. Kissin

"Zaporozhstal'" Plant

For a number of reasons the content of Fe, CaO and Mn in the sinter used at the "Zaporozhstal'" Plant is inconsistent and has fluctuations which are permitted by the technical conditions within the plant. In the quarterly cost accounting statements the content of Fe, CaO and Mn in the sinter is always mentioned.

To make allowance in the calculations between the sintering plant and the blast furnace for the sinter when it has deviations from the planned content of Fe and CaO, the planning section and the accounts section of the plant have developed and introduced a scale of allowances and addition payments, based on the following factors.

1. The change in the Fe content in the sinter compared with the planned value with constant content of limestone and other components of the charge is not connected with change in the planned consumption of raw material and can only be obtained due to change in the content of Fe in the sintered ore. In this case there is a change in the price of the sintered ore and consequently in the cost of the charge.

2. The change in the CaO content in the sinter is connected with change in the amount of limestone in the charge, which should be compensated by the amount of sintered ore. The quantity of other components of the charge remains as before. The content of CaO in the sinter can also vary with change in the consumption of lime. However, since the lime consumption is small (15-20 kg per ton of sinter), the content of CaO in the sinter is only corrected for limestone.

3. In accordance with the calculations of the accounts section of the plant it is assumed that the addition of 20 kg limestone to the charge gives an increase of 1% in the CaO in the sinter.

4. Starting from the condition of equality of the roasted solid residues, it is assumed that 20 kg of limestone in the charge of sinter is equivalent to 12 kg of sintered ore with an average moisture content of 5%.

Table 1 gives a calculation for the cost of the charge with varying contents of Fe and CaO in the sinter. The deviations from the basic content of Fe and CaO are taken as ± 1%.

As can be seen from Table 1, with the same content of CaO in the sinter, the change in the Fe content in it is caused by the change in its content in the iron ore, and the change in the CaO content in the sinter is due to change in the content of limestone and iron ore in the sintered charge.

Table 2 gives a calculation of the Fe content in the iron ore with a corresponding content of Fe and CaO in the sinter. The moisture content in the iron ore was taken as 4.9%; the amount of Fe introduced into the sinter by the other components of the charge is taken by calculation to be 59.7 kg per ton of sinter.

On this basis, calculations were made and presented in Table 1 for the amount of iron which should be introduced by iron ore for 1 ton of sinter.

Data are given below on the cost of the charge for 1 ton of sinter with varying contents of Fe and CaO and with deviations from the cost of the charge with the basic content of Fe and CaO.

Cost of charge rubles-kopecks

Content of Fe, %	With content of CaO, %		
	14.5	15.5	16.5
44.8	37-32	38-45	39-69
45.8	39-01	39-93	40-90
46.8	40-33	41-08	41-80

Deviations from the cost, rubles-kopecks

44.8	-2-61	-1-48	+0-24
45.8	-0-92	0	+0-97
46.8	+0-40	+1-15	+1-87

These deviations represent the scale of allowances and extra charges for the sinter. When calculating the cost according to this scale, the following factors are taken into consideration.

1. The base for the determination of the planned cost of 1 ton of sinter is taken as sinter with a content of 45.8% Fe and 15.5% CaO.

2. Sinter which does not meet the plant requirements for composition is valued at 25% less than the conditional price.

3. The allowances and extra charges are calculated by the sintering section from the average monthly analyses of the Inspection Department and are confirmed by the Head of the Planning Section of the plant.

4. If there is a change in the planned calculation for the basic content of Fe and CaO, the appropriate changes must be introduced into the scale.

An example is given below for the calculation of the extra charge for increased content of Fe and CaO in the sinter.

	Content, %	
	Fe	CaO
Basic	45.8	15.5
Calculated	46.01	15.68
Deviation	+0.21	+ 0.08

In the scale it is assumed that with a content in the sinter of 15.5% CaO and 46.8% Fe. i.e., 1% more than the basic value, the additional charge is 1 ruble 15 kopecks per ton.

Since in the case under consideration, the Fe content is only 0.21% greater than the basic value, we find the extra charge for 15.5% CaO and 46.01% Fe 1 ruble 15 kopecks × 0.21 = 0-24 kopecks.

We now find the extra charge for the same 0.21% Fe above the basic value with a content of 16.5% CaO:

a) for a 1% increase in the Fe content over the basic value with a content of 16.5% CaO, the additional charge is 1 ruble 87 kopecks– 0 rubles 97 kopecks = 0 rubles 90 kopecks;

b) for 0.21% Fe above the basic content with 16.5% CaO the additional charge is: 0 rubles 90 kopecks × 0.21

TABLE 1. Consumption of Components for 1 ton of Sinter and the Cost of Sinter of Differing Composition

	Amount of components, kg	Price for 1 ton, rubles-kopecks	Total rubles-kopecks	Amount of components, kg	Price for 1 ton, rubles-kopecks	Total rubles-kopecks	Amount of components, kg	Price for 1 ton, rubles-kopecks	Total rubles-kopecks
Content of CaO in sinter 14.5%									
Content of Fe in iron ore, %		53.6			55.0			56.4	
Content of Fe in sinter, %		44.8			45.8			46.8	
Iron ore	762	31—14	23—73	762	33—36	25—42	762	35—09	26—74
Manganese ore II s.	31	80—37	2—49	31	80—37	2—49	31	80—37	2—49
Carbonate ore	17	89—44	1—52	17	89—44	1—52	17	89—44	1—52
Flue dust	90	2—65	0—24	90	2—65	0—24	90	2—65	0—24
Pyrite cinders	50	18—50	0—93	50	18—50	0—93	50	18—50	0—93
Lime	13	139—44	1—81	13	139—44	1—81	13	139—44	1—81
Limestone	240	27—50	6—60	240	27—50	6—60	240	27—50	6—60
Total	1203		37—32	1203		39—01	1203		40—33
Content of CaO in sinter 15.5%									
Content of Fe in iron ore, %		54.4			55.8			57.2	
Content of Fe in sinter, %		44.8			45.8			46.8	
Iron ore	750	32—41	24—31	750	34—39	25—79	750	35—92	26—94
Manganese ore IIs	31	80—37	2—49	31	80—37	2—49	31	80—37	2—49
Carbonate ore	17	89—44	1—52	17	89—44	1—52	17	89—44	1—52
Flue dust	90	2—65	0—24	90	2—65	0—24	90	2—65	0—24
Pyrite cinders	50	18—50	0—93	50	18—50	0—93	50	18—50	0—93
Lime	13	139—44	1—81	13	139—44	1—81	13	139—44	1—81
Limestone	260	27—50	7—15	260	27—50	7—15	260	27—50	7—15
Total	1211		38—45	1211		39—93	1211		41—08
Content of CaO in sinter 16.5%									
Content of Fe in iron ore, %		55.4			56.8			58.2	
Content of Fe in sinter, %		44.8			45.8			46.8	
Iron ore	738	33—88	25—00	738	35—52	26—21	738	36—73	27—11
Manganese ore IIs	31	80—37	2—49	31	80—37	2—49	31	80—37	2—49
Carbonate ore	17	89—44	1—52	17	89—44	1—52	17	89—44	1—52
Flue dust	90	2—65	0—24	90	2—65	0—24	90	2—65	0—24
Pyrite cinders	50	18—50	0—93	50	18—50	0—93	50	18—50	0—93
Lime	13	139—44	1—81	13	139—44	1—81	13	139—44	1—81
Limestone	280	27—50	7—70	280	27—50	7—70	280	27—50	7—70
Total	1219		39—69	1219		40—90	1219		41—80

TABLE 2. Calculation of Iron Content in the Ore

Content of CaO in sinter, %	Weight of dry residue of iron ore, kg	Content of Fe in 1 ton of sinter, kg	Quantity of iron introduced in the sinter, kg			Weight of dry residue of iron ore, kg	Content of Fe in iron ore, %
			by other components		by iron ore		
14.5	762×0.951= =724.7	448	—59.7	=388.3	:724.7	=53.6	
		458	—59.7	=398.3	:724.7	=55.0	
		468	—59.7	=408.3	:724.7	=56.4	
15.5	750×0.951= =713.3	448	—59.7	=388.3	:713.3	=54.4	
		458	—59.7	=398.3	:713.3	=55.8	
		468	—59.7	=408.3	:713.3	=57.2	
16.5	738×0.951= =701.8	448	—59.7	=388.3	:701.8	=55.4	
		458	—59.7	=398.3	:701.8	=56.8	
		468	—59.7	=408.3	:701.8	=58,2	

= 0 rubles 19 kopecks; c) the total additional charge for 46.01% Fe and 16.5% CaO 0 rubles 97 kopecks + 0 rubles 19 kopecks = 1 rubles 16 kopecks. Thus, with a content of 46.01% Fe the additional charge will be: with a content of 15.5% CaO — + 0 rubles 24 kopecks; with a content of 16.5% CaO — + 1 ruble 16 kopecks, i.e. for 1% CaO above the basic value the additional charge will be 1 ruble 16 kopecks — 0 rubles 24 kopecks = 0 rubles 92 kopecks, and for 0.08% above the basic content of CaO the extra charge will be 0 rubles 92 kopecks × 0.08 = 0 rubles 07 kopecks

For an increase in Fe content by 0.21% and in CaO content by 0.08% the total additional charge to the sinter shop for 1 ton of sinter will be 0 rubles 24 kopecks + 0 rubles 07 kopecks = 0 rubles 31 kopecks.

This value is multiplied by the actual reduction of conditional sinter and is entered into the monthly cost accounts.

The allowances and extra charges for deviation of the manganese content in the sinter are calculated in the following way.

Starting from the planned charge, the planned content of Mn is established in the sinter (for a given period—1.46%). The manganese ore is planned for a content of Mn 36.0% and moisture content 19.5%.

In actual fact the content of manganese in the sinter averaged 1.40% for the financial month. Consequently, the deficiency of Mn for 1 ton of sinter

1.46—1.40 = 0.06% or 0.6 kg,

or for the whole production of sinter

400 000 × 0.6 = 240 tons. 0.805 = 828.5 tons,

Converted to manganese ore this comprises

240 : 0.36 : 0,805 = 828.5 tons,

the cost of which according to the price accepted in the plan is 80 rubles 37 kopecks per ton and will be an allowance with the cost of the total sinter which is allowed for in the cost accounts.

A CONFERENCE ON THE PLANS OF LARGE BLAST FURNACES

A Conference was held in Dnepropetrovsk from the 29th to the 31st of July to consider the plans for blast furnaces of 2000 m^3 volume and larger, developed by Gipromez.

The Conference was opened by N.I. Korobov, the deputy of the President of Gostroi in the Council of Ministers of the USSR. In his opening address he remarked that the June Plenum of the Central Committee of the Communist Party of the Soviet Union set high targets for the further improvement in production. As regards technical indices, the blast furnace workers of the USSR have beaten those of the USA. However, the average volume of US furnaces is 924 m^3 whereas that in the USSR is 844 m^3. It will be essential in the near future to construct blast furnaces of larger volume (2000 m^3 and above). They should be operated with improved methods. The fraction of fluxed sinter in the ore part of the charge should reach 100%, with an accompanying increase in the iron content; the blast should be heated to 1100—1200°. Natural gas and oxygen-enriched blast should find extensive application in blast furnace production. It is essential to put into production complex automation of furnace control, to increase the quality of refractories and introduce a number of other measures. This will lead to an over-all improvement in the technical equipment of blast furnace production and will also improve the technical and economic aspects of blast furnace operation.

Reports on the large furnaces were given at the Plenary Session by N.K. Leonidov (Engineering Section), I.M. Elinson (Equipment) and N.A. Ushakov (Constructional Section). These reports were discussed in detail at the Plenary and Branch Meetings.

The Gipromez plan for a 2002 m^3 blast furnace allows for two systems of operation: on blast with constant humidity, enriched with oxygen, with natural gas blasted into the furnace, and without natural gas with atmospheric blast of constant humidity. The temperature of the blast is 1200°, the gas pressure under the throat 1.5-1.8 atm. The furnace is planned to produce 4350 tons of

iron per day. The diameter of the hearth is 9750 mm, the body–10,900 mm. Its useful height is 29.4 m. Gipromez proposes 20 tuyeres for the furnace, 2 slag and 1 iron tapping hole. The hearth bottom is 5600 mm thick and is made up of carbon blocks and high alumina content bricks, the walls of the hearth are made of carbon blocks and the rest of the furnace is made of firebrick. The furnace is cooled by peripheral plate coolers.

The plan provides for water cooling except for the coolers under the hearth bottom which are air cooled.

The furnace is fitted with 4 air heaters, the heating surface of each being 27350 m^2. They are heated by a mixture of coke oven and blast furnace gas or natural and blast furnace gases.

The raw materials are fed to the skip pit by conveyers. The ground material from the sinter is screened below the bunkers, and is taken by special conveyers to the skip hoists and loaded into the cars. The large sinter is taken by the main conveyer to the weighing funnel from where it enters the skip. The coke from the bunker is taken by feeders to the screen to remove the fines and then to the coke scales.

The plan allows for the automation of the collection and loading of the charge, stabilization of the heat and the humidity of the blast and controlling the heating of the air heaters. Later it is proposed to establish a system of automatic control of the blast and the gas at the tuyeres and to control the furnace operation by the drops in static pressure.

The furnace equipment will be modernized. The furnace will be provided with a special machine for splitting the iron tapping hole. Mechanized troughs are used for pouring the iron and slag.

The plan allows for the assembly of the blast furnace in large units. Extensive use will be made of prefabricated reinforced concrete constructions.

The reports were discussed by production workers, representatives of planning organizations, research institutions and teaching institutes. Those taking part in the discussion provided many valuable corrections for the plan. Serious discussion was given to the profile of the furnace.

There was a good deal of controversy on the problem of raw materials for the blast furnace smelting. Large blast furnaces should operate with well-graded and fractionated ore.

Lively discussions took place on the problem of the number of iron tapping holes for the furnace.

At the section dealing with equipment a discussion was held on the machine for dismantling the tapping hole and on an electrical drum. The discussions on the design of the pouring apparatus were particularly involved. The blast furnace workers and representatives of the planning organizations provided much interesting and useful information based on experience at the plants.

At the Plenary Meeting there was a careful discussion of the plan decided on by the conference. It was decided to recommend blast furnace operation with blast enriched with 35% oxygen. The useful height of the furnace was taken to be 29.4 m. It was decided to use plate coolers for the shaft. It was decided that Gipromez should develop a system with cantilever coolers. It was also decided that Gipromez should develop a system with two iron tapping holes.

A proposal was accepted that the possibility be left open of converting the large volume blast furnace to evaporation cooling. It was also decided to develop a standard design for a blast furnace for operation with ores containing zinc. The blast temperature was taken as 1200°. To provide this temperature a standard was laid down for the heating area of the air heaters, equal to 80 m^2 per 1 cubic meter of the furnace. There are to be 4 air heaters in the block of apparatus servicing the furnace.

The conference decided on the desirability of developing a plan for a blast furnace of greater volume than 2000 m^3 and laid down the main ways for developing the plan.

The basic dimensions of equipment for such a furnace would be based on a maximum volume of 2700 m^3.

REMOVAL OF CARBON FROM THE MELT AT LOW CARBON CONTENTS

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For generalizing the results of blowing the bath with oxygen when melting low-carbon steels, particularly transformer steels, chronometer-controlled melts were made in an electric arc furnace, the oxygen being introduced into the bath by various methods.

The boil finished with a content of 0.06–0.35% C in the metal, and oxygen was then blown. The oxygen (purity 95–97%, pressure 6–8 atm gauge) was fed to the bath through iron lances 3/4 inch and 1 inch in diameter, lined with a special composition (magnesite and water glass), or by a combined method, using a water-cooled five-jet tuyere through the roof, and lined iron lances through the door. The metal was blown with the current switched off and the electrodes raised. At the end of blowing, the temperature of the metal varied from 1615 to 1710°C, as indicated by a tungsten-molybdenum immersion thermocouple.

During blowing, samples of the metal were taken and deoxidized by means of aluminum. Special attention was paid to ensure that the samples were clean. After cooling them in clean water, the surface was removed on an emery grinder to a depth of 0.8 mm, after which drillings were taken for analysis. The carbon content was determined by the Würtz method using a small burette (for 0.25% C) with a scale division value of 0.001%. The results obtained were thus sufficiently accurate.

As will be seen from the accompanying table of results, the rate of carbon removal can be grouped according to carbon content of the bath: up to 0.08%, 0.08–0.06%, 0.06–0.03%, and 0.03–0.02%. Up to a content of 0.08 % C, the rate of carbon removal depends largely on the quantity of carbon at the commencement of blowing and varies from 0.5 % C/hour for C_{init} = 0.11–0.12% to 1.15 % C/hour for C_{init} = 0.35%.

The table shows that the temperature of the metal in this period does not have any significant influence on the rate of oxidation of the carbon. This is due to the relatively high assimilation of oxygen which, according to foreign data, is 100% for a content of 0.40 %C and above and is only 2% for 0.025% C.

The high oxygen consumption in oxidizing the carbon is evidence of the fact that the rate of carbon removal also depends on the continuity of oxygen supply to the bath. In one of the melts (No. 30909), as a result of interruptions in blowing due to the necessity for changing the lances, the rate of carbon removal dropped almost to half.

In the range of contents from 0.08–0.06% C, the rate of carbon removal is practically independent of the initial carbon content, being 0.3–0.5% C/hour. In the oxidation of carbon in the range 0.06–0.03% C, a sharp drop was observed in the rate of carbon removal down to 0.014–0.26% C /hour. The rate of carbon removal v_C = 0.36–0.52 % C/hour only in melts with a high carbon content at the commencement of blowing; in a melt with discontinuous blowing, v_C = 0.09 % C/hour. In the oxidation of carbon with 0.03 to 0.02%, the rate of carbon removal was from 0.100 to 0.033 % C/hour.

On the average, in the carbon content range 0.08–0.02%, the rate of carbon removal was from 0.112% C/hour (duration of blowing 32 min) to 0.248 % C/hour (14.5 min). In the case of discontinuous blowing, v_C = = 0.087 % C/hour, and the blowing period was 41 min.

It is interesting to note that in one series of melts, due to delay in receiving the analysis of the metal samples, blowing was carried out at a content of 0.02% C. A bath carbon content below 0.018–0.019 % was never even once obtained. The fact that the carbon content did not change after stirring the bath with iron rabbles and while the slag was being run off shows that the carbon content was approximately the same throughout the whole of the metal (in any event, the difference did not exceed a few thousandths percent).

In addition to the influence of the initial carbon content and continuity of blowing on the rate of carbon removal, an important part is played by the rate of oxygen consumption. In this connection, it is necessary to consider the means of introducing the oxygen, since parameters such as pressure and purity of the oxygen are not always maintained regularly in production conditions. For an oxygen pressure in the supply pipeline of 8 atm, the consumption of oxygen when a 3/4 inch diameter lance was used was about 750 m^3/hour, with a one inch diameter lance it was 870 m^3/hour and with a tuyere it was 750 m^3/hour.

It is known that other conditions being equal, the time required for the removal of carbon from the metal from 0.25 down to 0.03–0.02 % C can be almost halved by increasing the oxygen consumption from 700–900 to 1680 m^3/hour. To a considerable extent, the rate of oxygen consumption determines the rate of oxidation of

TABLE. Principal Data on the Blowing of Metal with Oxygen

No. of melt	Method of introducing the oxygen	Carbon content at the commencement of blowing, %	Oxygen pressure, atm. gauge	Purity of oxygen, %	Clear blowing time, min.	Consumption of oxygen for blowing			Rate of oxidation of carbon, % C/hour					Temperature of metal at the end of blowing, °C
						total, nm^3	for 0.01% C, m^3	per minute, m^3/min	0.08% C	0.08—0.06% C	0.06—0.03% C	0.03—0.02% C	Average rate in the interval 0.08-0.02% C	
30803	Lance d = 3/4 inch	0.08	11.0	95.0	32.0	501	83	15.6	—	0.40	0.14	0.05	0.112	1690
30841	The same	0.12	8.0	95.0	31.0	485	48.5	15.6	0.56	0.40	0.22	0.04	0.138	1710
31082*	" "	0.08	6.0	96.0	25.5	350	58.0	13.5	—	0.30	0.225	0.043	0.138	1630
30760	" "	0.23	8.0	95.0	34.0	440	25.6	15.9	0.95	0.48	0.26	0.050	0.147	1650
30749	" "	0.35	7.5	95.0	29.0	463	14.0	16.0	1.15	0.60	0.36	0.075	0.163	1600
30849	Lance d = 1 inch	0.08	8.0	96.0	24.0	405	68	16.9	—	0.34	0.21	0.05	0.15	1620
30932	The same	0.11	7.0	95.0	33.0	540	60	16.3	0.51	0.40	0.21	0.035	0.122	1620
30909*	" "	0.27	8.0	95.6	58	855	34	13.3	0.52	0.22	0.09	0.037	0.087	1670
30920	" "	0.34	7.0	95.0	36.0	580	18	16.1	1.15	0.60	0.51	0.035	0.160	1700
30963	Tuyere and lance, d = 1 inch	0.06	8.0	96.0	23.0	475	120	20,6	—	—	0.18	0.046	—	1615
31069	Tuyere and lance d = 3/4 inch	0.13	6	97.0	24.0	400	36.6	16.7	0.70	0.53	0.21	0.067	0.183	1630
31104	The same	0.13	7	96.0	17.0	350	32	20.5	1.20	0.80	0.26	0.100	0.248	1645
31135	" "	0.15	6.5	96.5	21.5	400	30.8	18.6	0.67	0.54	0.278	0.092	0.240	1600

*There were interruptions in blowing

the carbon. This is particularly noticeable for a content of 0.03—0.02%C when the bath is blown with two lances. On the average, in melts where the metal was blown for 1—2 min through two lances (with change of lances), v_C = 0.05% C/hour. At the same time, increasing the oxygen consumption to 1400 m^3/hour and blowing all the time through two lances does not permit intense overheating of the metal in the vicinity of the front bank.

According to published data, oxidation by oxygen of 0.01% C results in an increase in bath temperature of 3.5°C, and oxidation of 0.01% Fe gives an increase of 3.3°C. In the melts described, oxygen-blowing of the steel increased the temperature of the metal by 60—100°C. Introduction of oxygen by two lances increased the local heating of the steel 1.5 to 2 times, resulting in destruction of the furnace lining. Due to this, at the suggestion of V.P. Frantsov, we tried blowing the metal with oxygen using a one inch diameter lance, enabling the oxygen consumption to be increased without overheating of the bath.

Still better results are obtained by combined introduction of the oxygen through a five-jet water-cooled tuyere, with whirling of the oxygen jet, and a 3/4 inch diameter lance. The oxygen consumption per minute increased sharply and the blowing time was reduced by 8—10 min, due to the increased rate of carbon removal. At the same time, due to the fact that the oxygen was introduced into the bath at different points, no cases of overheating of the metal with the accompanying destruction of the furnace lining were observed. It should be mentioned that these data relate to working with oxygen at a pressure of 6—8 atm gauge pressure.

This combined blowing has currently been introduced into industry. Removal of the slag before oxygen blowing plays a positive part in increasing the rate of carbon removal. As a result of the work which has been carried out, the following conclusions may be drawn:

1. The use of oxygen for oxidizing the carbon in the melting of low-carbon steels appears to be essential; the purity of the oxygen should not be less than 95% (the use of commercially pure oxygen is desirable).

2. The rate of carbon removal when the metal is oxygen blown depends mainly upon the rate of introduction of the oxygen (m^3/hour) and on the original carbon content of the bath. In the range from 1600—1700°C, the temperature of the metal was not found to have any influence on the rate of carbon removal.

3. The optimum blowing conditions in electric furnaces of medium capacity are: Combined introduction of oxygen (through the roof by means of a water-cooled tuyere and through the working door by means of a lined lance); temperature at the commencement of blowing about 1570°C, initial carbon content 0.10—0.15%. Before blowing, the slag should be run off. These conditions ensure oxidation of the carbon down to 0.02% in 20 min without overheating the bath.

THE CONVERTER SHOP OF THE KRIVOI ROG PLANT

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The converter shop of the Krivoi Rog Steel Plant was designed for the iron to be blown by pure oxygen from the top through a special water cooled tuyere.

After the iron is poured in, the converter is charged with ore and lime. The consumption of ore is 4.5%, lime 8-9%, bauxite 0.8-1%. The metal is poured from the converter through a tapping hole at the top and the slag is poured through the top.

The converters which have been installed have a volume of 44 m^3 and the following dimensions, mm:

Internal diameter	3480
Height from the bottom to the top edge	5335
Internal diameter of top	1830
Thickness of lining (total)	.455 and 510

The converter is lined with chrome-magnesite brick. The iron is poured into the converter from a ladle arranged on a trolley (Fig. 1). The additions and the pouring of iron are by remote control. The shop has two 1300-ton mixers. In the mixer building there are two casting cranes of capacity 125/30 tons and a machine for removing slag from the ladles. The shop has an apparatus for trapping graphite which is liberated when iron is poured from the mixer and when slag is removed from the ladle.

The ore, lime, bauxite and ferromanganese are in the charge bay, equipped with two magnetic grab cranes of capacity 10/10 tons and grab capacity 1.5 m^3.

To prevent the lime forming dust in the lime roasting shop, it is charged into containers of 3.5 m^3 capacity, and transported to the charge yard on railway platforms. Here the containers are put onto a special platform or directly onto the 20-ton electrical locomotive for feeding to the converters. The ore and bauxite at the charge yard are loaded by grab crane through an intermediate bunker of capacity 26 m^3 into containers of capacity 2 m^3 and are fed to the electrical locomotive.

Between the two converters at a level of 25.6 m there is a 5-ton bridge crane which takes buckets from the electrical locomotives and loads them into bunkers which are at a level of 22.7 m. Each converter has 3 bunkers: ore (capacity 15 m^3), lime (22 m^3) and bauxite (15 m^3). Under the bunkers there are belt feeders which feed the required amount of material into the weighing funnel which weighs up to 5 tons. This funnel has a gate, and when this gate closes the material passes along a trough into the converter. The bunker gates are mechanized.

After the materials have been charged and the iron has been poured the converter is raised and the tuyere is lowered at a rate of 0.25-0.50 m/sec into the top. At the same time, oxygen is fed in, its specific consumption being 60 m^3/ton of steel; the oxygen pressure in the system is 8-12 atm. The water for cooling the tuyeres is fed in at a pressure of 9 atm in quantities of 60-70 m^3/hr. In the oxygen pipe near the converter there are two valves, one of them is a cut-off valve and the other is a control valve.

To remove the gases which are liberated in the converter, at a distance of 0.5 m from the top there is a water cooled caisson of diameter 2.8 m, length 7.3 m, which can be removed from the top of the converter by means of a special mechanism. The water flow at the caisson is 300-400 m^3/hr at a pressure of 3 atm. The caisson is connected with a lined metal pipe. Fig. 2 shows an over-all view of the converter and caisson.

The internal diameter of the gas pipe is 3.94 m. In its upper part the gases are returned to a water cooled part of the chimney where they are cooled to 80-100° by water fed by sprays, they then pass on to a checkerless scrubber of diameter 5 m, height 20 m, from which they enter venturi pipes with a drop in pressure of about 400 mm water gauge and into a cyclone of diameter 3.6 m. After scrubbing, the gases are expelled by an exhaust fan into a 65 m pipe. The content of dust in them is about 150 mg/m^3 with an initial contamination with dust of about 80 g/m^3. The delivery rate of the exhaust fan is 75000 m^3/hr, pressure reduction 1200 mm water gauge, motor power 500 kw. The temperature of the gases at the exhaust fan is 35°. Each converter is fitted with gas scrubbing.

The charging of the converter is entirely mechanized. The tilting of the converter, the lowering and raising of the tuyere and also the gas scrubbing at each converter are appropriately interlinked. These operations are controlled by an operator.

The whole system for interlinking and controlling the feed of oxygen and water is equipped with control and measuring apparatus which not only shows but also signals reductions in the quality of oxygen and the pressure of water. At one of the converters there is a boiler for utilizing the heat and electrical gas scrubbing.

In the casting bay there are 3 railway tracks; on 2 of them there are 4 teeming platforms with stands for pouring metal. The bay is equipped with 4 casting cranes of capacity 75/15 tons.

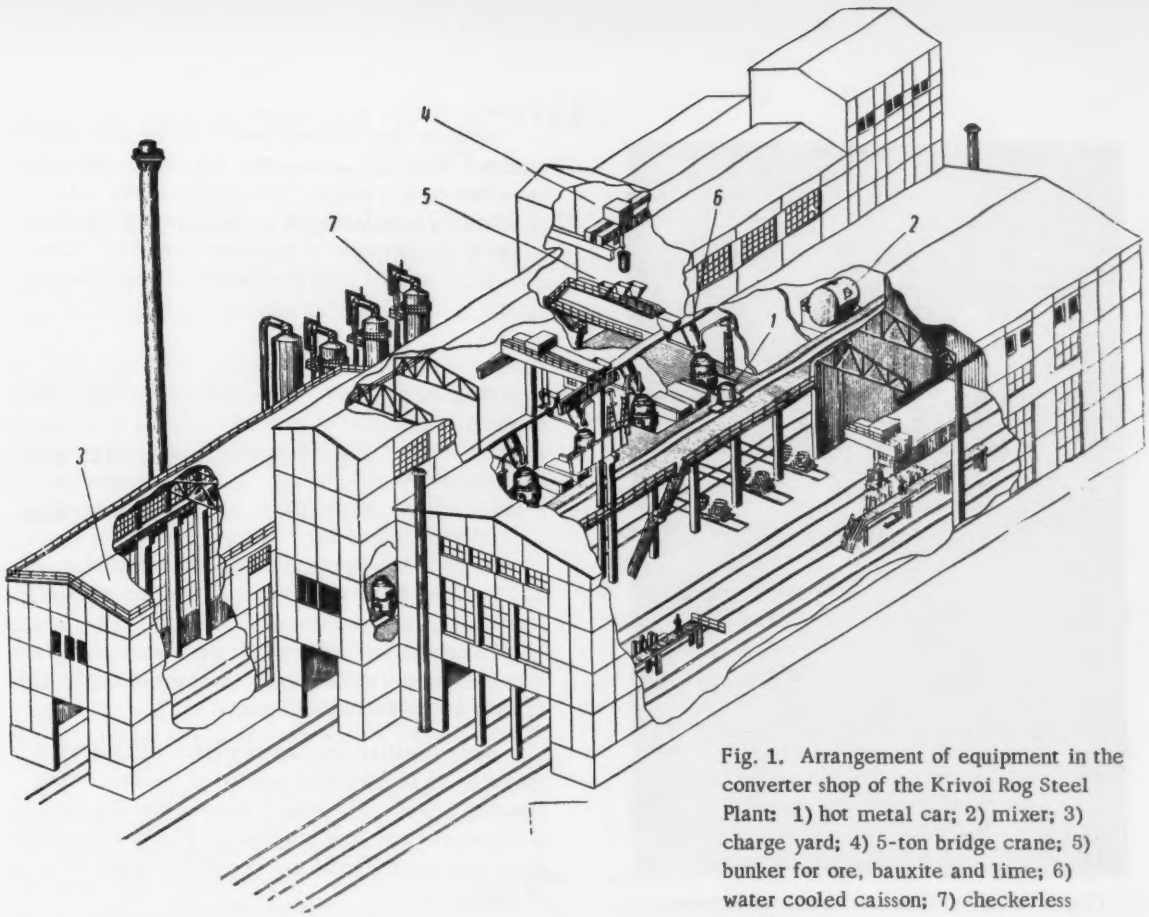

Fig. 1. Arrangement of equipment in the converter shop of the Krivoi Rog Steel Plant: 1) hot metal car; 2) mixer; 3) charge yard; 4) 5-ton bridge crane; 5) bunker for ore, bauxite and lime; 6) water cooled caisson; 7) checkerless scrubber.

The metal from the converter is poured into a steel pouring ladle fixed on an electrified trolley. The movement of the trolley is controlled from a panel below the working platform. The casting cranes take the ladles with the metal to the stands for teeming. The trolleys with the molds are placed under the ladle by means of a pusher. The slag from the converters is poured into slag ladles of 11 m^3 capacity. The slag trolley is connected by an automatic coupler to the hot metal car. The ladles filled with slag go to the slag section which is near the main body of the shop.

At the end of the converter section there are special stands: for tamping the linings of the converters (with drive), for putting in new lining and for drying. The feeding of brick and solution and also the drying are mechanized. The converter is taken to the stand for repair and returned by a 150/30 ton bridge crane. Arranging the repairs at some distance from the working converters has enabled the output of the shop to be increased by about 25% and has set up favorable conditions for the repair and operation of the converters.

In the casting bay there are two compartment kilns for drying the stoppers and stands for installing the nozzles and stoppers into the ladles. The furnaces and stands are serviced by two 2.5 ton cantilever cranes.

After casting and soaking, the ingots are sent from the casting bay to the section where they are stripped, this section being equipped with two bridge cranes of 25/15 ton capacity and 175 tons pushing power. The molds are cleaned by water from two sprays (two molds at a time) at a pressure of about 70 atm and are then sent to a section where there is a stationary machine for lubrication. The mold yard is equipped with 4 cranes of lifting capacity 20/5 tons and has 4 platforms for preparing sets of molds and also 3 railway tracks.

In the first place the oxygen station was planned in the form of 3 KT-3600 units with appropriate air and oxygen compressors. At the present time there are two BR-5 units, each delivering 5000 m^3/hr.

In the lime roasting shop there are 4 shaft furnaces of up to 141 m^3 volume, working on gas. The roasted lime enters the grading section after which it is loaded into containers and sent to the converter shop. The gases from the lime roasting furnaces pass through a gas scrubber and are then put into the atmosphere through a pipe.

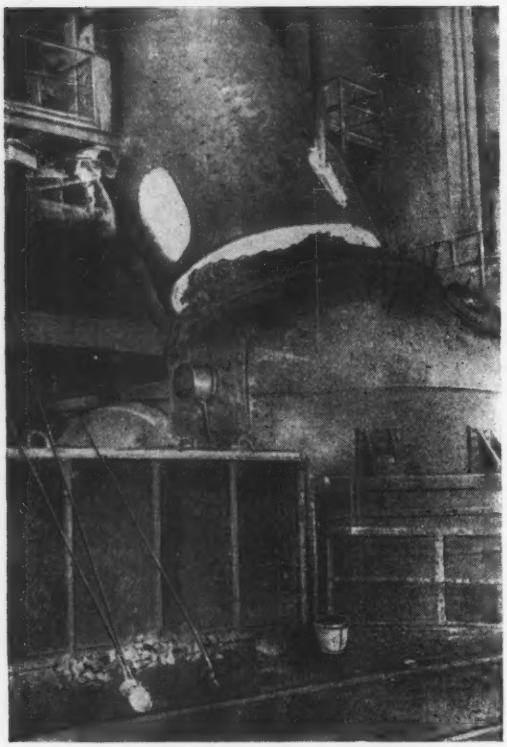

Fig. 2. Over-all view of converter and caisson.

The technique of smelting steel. Iron of the following composition was used for blowing, %:

Si	Mn	S	P
0.5–0.8	1.0–1.4	up to 0.055	up to 0.12

After the iron has been poured in, the converter is arranged in the vertical position and the ore and lime are added from a trough. The quantity of ore is determined by calculating for a final temperature of the metal in the converter of not more than 1630°. The quantity of ore and lime are found from the following data:

Content of Si in the iron, %	0.4	0.5	0.6	0.7	0.8	0.9
Quantity of ore, kg	1550	1700	1850	2000	2100	2300
Quantity of lime, kg	1800	1950	2050	2250	2350	2450

For each 100 kg of ore added it is essential to add 50 kg of lime in excess of the indicated amount (to form a slag with the silica of the ore).

The metal is blown with a tuyere having a 65 mm diameter nozzle. The oxygen consumption is 120-130 m^3/min at a pressure of 6-8 atm.

When smelting low carbon rimming steels, there should be no intermediate pouring of the slag. When smelting killed steels, after 7 minutes blasting the slag is tapped and and lime is added for a second time after which the blasting continues to give the required content of carbon.

Carbon steels are smelted in the converter shop of the Krivoi Rog Plant. In its chemical composition the converter metal answers the requirements of GOST 380-50 for open-hearth steel. The nitrogen content averages 0.006 %.

PROTECTING NEW MOLDS FROM OXIDATION DURING STORAGE

A.M. Maslov

At many steel plants new molds are subjected to "aging" or lengthy storage before they are used, which means that rust can be formed on the working surfaces of the molds to a depth of 2-3 mm. This rust can be removed either by washing the molds with molten slag or by cleaning with brushes. These methods, however, do not always prevent breakage when the first ingot is cast.

At the Il'ich Plant the author suggested that, after cleaning and blasting the molds with compressed air, they should be dressed in the casting shops with dehydrated tar. This protection preserves the internal surfaces of the molds from oxidation for an indefinite length of time, regardless of the change in the temperature and atmospheric conditions.

DROP IN TEMPERATURE OF THE STEEL DURING CONTINUOUS POURING

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The drop in temperature of the metal during continuous pouring considerably complicates the pouring conditions of a melt. The drop in temperature of liquid steel after tapping depends on the quantity of the metal and its temperature before tapping, the duration of tapping, the form and length of the tapping spout, the form, dimensions and temperature of the lining of the steel ladle and the intermediate pouring vessel, the physicochemical properties of the refractories, the amount of slag in the ladle, etc.

The heat losses of the metal after tapping from the furnace have been studied at both Russian and foreign plants. Thus, at one foreign plant, where continuous pouring of steel was carried out from a 7-ton ladle, after transferring steel to it from a 70-ton ladle, it was found that the total drop in temperature of the metal from the moment of tapping to entering the mold was 85–95°C the drop in temperature of transferring from the large to the small ladle being 40–50°C. The drop in temperature of the metal in the ladle depends largely on the duration of tapping. For a tapping time (60 tons) of 3 min, the temperature of the metal in the spout fell by 30°C, and for a tapping time of 7 min by 80°C.

The total drop in temperature of the metal for an 80 ton melt from the moment of tapping to the commencement of pouring is 110–115°C, of which 30–35°C is lost in the spout. In a study of the heat losses in a 60-ton ladle, it was found that the difference between the temperatures of the surface and bottom layers of the metal was 70–100°C.

For determining the heat losses in the continuous pouring of steel, a study was made of the process of pouring from a 50-ton ladle by means of a two-nozzle intermediate pouring vessel having a capacity of 3.5–4 tons and a nozzle orifice diameter of 30 mm. Before pouring commenced, the intermediate pouring vessel, closed with a lined cover, was heated by means of an oil burner for 40–50 min to a temperature of 800–1000°C. The temperature of the metal was measured by means of tungsten-molybdenum immersion thermocouples in the furnace before tapping, in the 50-ton ladle and in the intermediate pouring vessel during pouring.

From temperature measurements made in 100 melts, it was found that the most characteristic drop in temperature between the metal in the furnace and in the intermediate vessel was 115–120°C. This drop is characteristic of current pouring conditions: tapping the steel into an unheated ladle, inadequate heating of the intermediate vessel (800–1000°C) and uncovered portions of metal surface in the region of the stoppers in the intermediate vessel. A further study of the working of the installation showed that the minimum steel temperature in the intermediate vessel at which it was possible to pour smoothly to the end was 1530–1540°C. In this way, it was found that the optimum temperature of the metal in the furnace before tapping for 50-ton melts poured in the continuous installation was 1640–1660°C.

A study was also made of the temperature distribution in the steel ladle and the variation in temperature of the metal in the ladle during pouring. The ladle was provided with a second "false" stopper containing tungsten-molybdenum thermocouple leads insulated by a porcelain tube (Figs. 1 and 2). The thermocouple junctions were brought to the outside through orifices in the stopper sleeves and protected by alundum caps with a zirconia luting. The temperature of the metal in the ladle was measured at three points distributed vertically at the following distances from the ladle bottom: 200–250, 800–900 and 1625 mm. At the same time, the temperature of the metal in the intermediate pouring vessel was measured by means of a floating tungsten-molybdenum thermocouple.

After tapping, the ladle with the steel was taken to the working platform of the continuous pouring installation where the thermocouples of the "false" stopper and the floating thermocouple in the intermediate pouring vessel were connected to a recording instrument by means of a switch. Thus the first measurement of the temperature of the steel in the ladle was made only 20 min after the commencement of tapping. The variation in temperature at the different points in the steel ladle and in the intermediate pouring vessel is shown in Fig. 2. The thermocouples in the ladle gave reliable readings for an average of 10–15 min. The failure of the top thermocouple was evidently due to attack of the cap by slag as the level of the metal fell in the ladle. Certain fluctuations in the temperature of the metal at a given level are evidently due to movement of the metal in the ladle during tapping and also to the descent

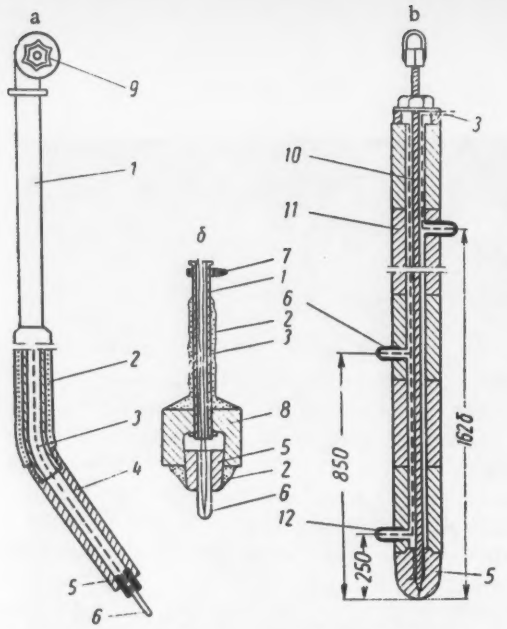

Fig. 1. Diagram of thermocouples used in the investigation. a) Immersion thermocouple used in furnace; b) floating thermocouple; three-point thermocouple of the "false stopper"; 1) sleeve; 2) luting; 3) thermocouple leads; 4) graphite block; 5) plug; 6) cap; 7) collar; 8) fireclay float; 9) reel with supply of electrodes; 10 rod; 11) stopper sleeve SP-6; 12) junction T-R6.

of differently heated layers during pouring. Fig. 3 shows the vertical temperature distribution in the steel ladle.

These experiments showed that there was a considerable drop in temperature in the vertical direction of the ladle, attaining 70°C in the present case. These data are in agreement with the investigations mentioned in the introduction to this article and reveal the considerable heat losses in the main ladle after it has been filled.

Due to the convection currents—movements caused by the difference in density of the differently heated layers—a distribution of the differently heated portions of metal occurs vertically in the ladle. In the bottom part of the ladle are the first portions of metal from the lower, colder zones in the furnace with additional cooling in the spout and along the walls of the ladle. This metal also loses much heat by radiation from the uncovered surface.

After the ladle has been filled, the metal continues to cool at the walls and the cooled portions descend to the bottom. Thus, in the first 8–10 min after filling there is evidently a redistribution of the differently heated portions of metal in the vertical direction of the ladle. It was found that at the commencement of pouring, the lower, cooled portions of metal enter the intermediate pouring vessel. Some drop in temperature of the metal occurs in the stream in the transfer from the main ladle to the intermediate vessel. Temperature measurements in the stream and in the metal in the intermediate vessel show that this drop was 7–15°C.

Tests showed that during pouring, in the 4 ton intermediate vessel, the temperature of the metal fell by 25–40°C, depending on the heating of the metal and of the lining surface.

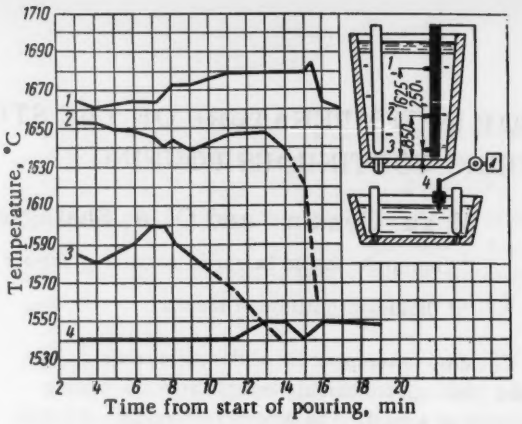

Fig. 2. Variation in temperature of the metal in the intermediate pouring vessel and ladle during the pouring process (tapping time 3 min).

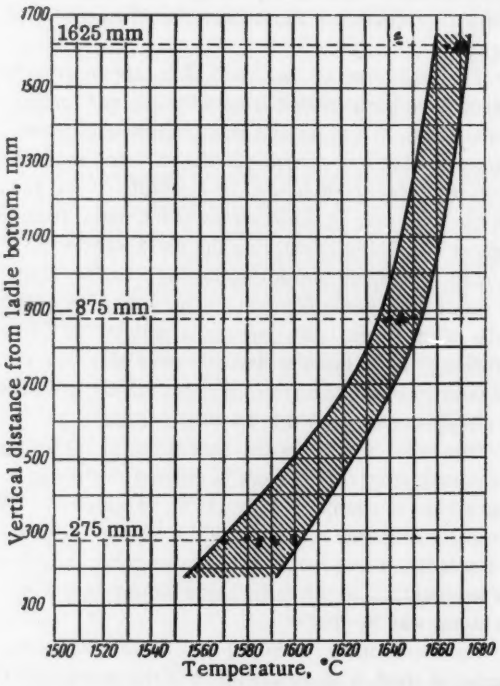

Fig. 3. Vertical temperature distribution of the metal in steel ladle (time of measurement 10 min after commencement of tapping).

Temperature of Metal at the Various Measuring Points

Measuring point	Temperature of metal (°C) at time of measurement from moment of tapping the melt, min															
	0	5	15	37	40	65	0	7	14	27	49	0	10	14	22	31
	I- Duration of tapping 3 min						II- Duration of tapping 2.5 min					III- Duration of tapping 3 min				
In furnace, before tapping	1700						1655					1670				
In steel ladle		1655						1630					1650			
In stream from steel ladle				1600		1585				1570	1580				1600	
In intermediate vessel			1590			1570			1575	1560	1570			1590	1587	1585
In stream from intermediate vessel					1540						1535					
In mold					1530					1530				1550		

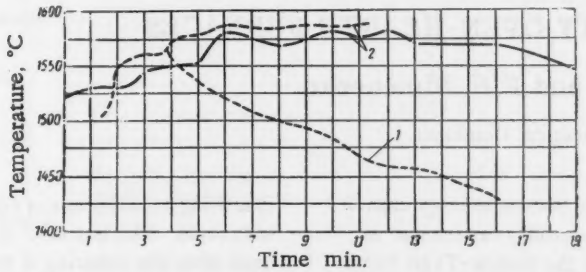

Fig. 4. Cooling of the first portions of metal in the intermediate pouring vessel. 1) In the nozzle well; 2) at the stopper.

In three melts, the temperature was measured successively: in the furnace before tapping, in the ladle, in the stream from the ladle and in the mold (see table). The data presented in the table are in good agreement with the indicated drop in temperature in the trough, ladle and intermediate pouring vessel.

The drop in temperature of the first portions of metal in the intermediate vessel was measured by means of tungsten-molybdenum thermocouples inserted in the lining, the junctions of which were led to the surface in the region of the nozzle wells, and were protected by alundum caps with magnesite luting. As will be seen from Fig. 4, the first portions of metal in the intermediate vessel are cooled considerably, and this represents a considerable danger for the normal pouring of the melt. In a number of cases where the metal in the intermediate vessel had a temperature below 1540°C, as shown by a tungsten-molybdenum immersion thermocouple the melt could not be poured; a crust was formed on the nozzle neck, the stopper stuck or the metal solidified in the nozzle passage while the first few portions of metal were being poured.

Experiments also showed that the temperature of the metal in the intermediate vessel does not fall during pouring, but in some cases, a certain rise in temperature was observed toward the end of pouring. These results confirm the accuracy of the conclusions regarding the uneven vertical temperature distribution of the metal in the steel ladle.

At the same time, heating of the metal to a higher temperature does not improve conditions for continuous pouring. An analysis of the working statistics of the continuous pouring installation for 6 months of 1958 showed that a higher steel temperature had a negative effect on the life of the stoppers, increased the stickers in the molds and increased the total amount of ingot scrap.

Heating of the steel ladle is a radical means of reducing the heat losses in the metal during continuous pouring. In one foreign plant, in the investigation of the heat losses, it was found that in filling a 20-ton ladle heated to 850°C, the temperature of the metal fell by 15–20°C, but in filling the same ladle heated to 200°C, the temperature of the metal fell by 35–40°C. In the practice of another plant, pouring the metal in a continuous pouring installation, the main and intermediate ladles were heated to 1470–1520°C, which made it possible to pour the metal at a lower temperature (on the average 20°C lower) than with ordinary pouring.

The following conclusions may be made on the basis of the work which has been carried out:

1. A considerable drop in temperature occurs in the steel ladle. In a 50-ton ladle, the temperature of the metal drops by 10–100°C. For reducing the total heat losses, it is preferable to preheat the steel ladle before tapping.

2. In the intermediate pouring vessel of the continuous pouring installation, cooling of the metal by 25–40°C occurs, and this cooling depends primarily on the degree of preheating of the intermediate vessel and also on the area of the contact surface and thermophysical properties of the refractory materials.

3. In the pouring process, the metal enters the mold at practically constant temperature; sometimes a rise in temperature of the metal is observed toward the end of pouring, due to the descent of upper, hotter portions of metal.

4. The first 3–5 min constitute the dangerous time for continuous pouring, when the most-cooled portions of metal are passing through the nozzle of the intermediate vessel. This gives rise to the necessity for heating the steel pouring nozzles in the intermediate vessel before and during pouring.

RAPID SINTERING OF THE NEW BOTTOM IN LARGE-CAPACITY OPEN-HEARTH FURNACES

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Nizhne-Tagil Metallurgical Combine

A rapid sintering of a new bottom in large capacity open-hearth furnaces was successfully carried out at the No. 1 open-hearth shop of the Nizhne-Tagil Metallurgical Combine at the end of 1958. Magnesite powder, grade "Ékstra", ground open-hearth slag from rimmed-steel heats and dry scale from rolling mills were used (Table 1).

The rapid sintering of the bottom is quite different from the previous methods of sintering. Comparative data on the rapid and ordinary sintering is given in Table 2.

A distinguishing feature of the rapid method of bottom sintering is that it takes a much shorter time; 39 hours 47 minutes, compared with 147 hours 25 minutes for sintering by the ordinary method. This big difference in time is explained by the fact that with the rapid method only 4 layers, instead of 14, are sintered in; in addition, some other modifications have been introduced.

The object of this method is to obtain from the first three layers a base which will adhere firmly to the brickwork of the furnace bottom and then to form a dense heat-resisting top crust from the thick fourth layer consisting on the bottom of magnesite powder and covered with scale. The last layer is formed during the first heats after the sintering of the bottom.

After observing the temperature of the bottom when it had remained constant for eight hours, we started slagging the brickwork of the bottom. The slag was charged from the fettling machine at 15-20 minute intervals on the back and front walls and on the banks from which excess slag flowed onto the bottom. Previously, while the bottom was sintered with approximately the same quantity of slag, the latter was charged with the fettling machine in five stages with 1-3 hour intervals. At the end of the slagging, the bottom was covered with scale.

The first sintered layer from the magnesite powder was spread over the whole surface of the bath. It was then heated for 7 hours 40 minutes. The material for the second and third layers consisted of magnesite powder, slag, and scale, whereas previously no scale was added. The material for the second layer contained 12% slag and 3.5% scale, and for the third layer–21% slag and 4% scale.

TABLE 1. Chemical and Screen Analysis of Materials

Material	Chemical composition, %								Screen analysis (remaining on the screen, mm), %						
	SiO_2	Al_2O_3	CaO	MgO	MnO	FeO	Fe_2O_3	Fe	8	5	3	2	1	0.4	below 0.4
Magnesite powder . .	2.64	1.04	2.02	91.5	—	—	2.16	1.51	—	15.35	23.0	22.95	25.9	5.95	6.85
Ground slag	14.0	7.73	39.15	16.64	6.0	8.64	3.96	9.49	1.78	—	10.85	36.85	32.95	5.25	14.1
Scale	—	—	—	—	—	59.18	38.94	73.24	—	—	—	—	—	—	—

TABLE 2. Operating Statistics on the Sintering of the New Furnace Bottom

Item	Sintering method	
	rapid	ordinary
Duration of heating (moment of starting the gas flow through the checkers), hours-minutes . .	68–00	84–15
Total duration of sintering, hours-minutes. . . .	39–47	147–25
Slagging		
Duration, hours-minutes	9–05	21–55
Consumption, kg:		
slag.	34 200	36 220
scale.	2 070	2 600
Sintering proper		
Duration, hours minutes	27–25	118–25
Consumption, kg:		
magnesite powder	40 210	57 021
slag.	1 608	13 238
scale.	1 556	–
Slagging the bottom		
Duration, hours-minutes	3–17	7–05
Consumption kg:		
slag.	–	14 000
scale.	30 000	14 000
Consumption of fluxing materials (slag and scale) for the sintering proper and for slagging of the finished bottom, kg.	33 164	41 238
Total consumption, kg	109 644	137 079
consisting of:		
magnesite powder.	40 210	57 021
slag.	35 808	63 458
scale.	33 626	16 600
Consumption of fluxing materials in % of the total consumption, %:		
without taking into account the slagging of the brickwork (for bottom formation)	45.2	42.0
taking into account the slagging	63.3	58.4
for the sintering proper	7.2	18.9
Fluxing materials absorbed by the bottom, kg . .	56 434	70 058
Number of layers:		
basic.	4	14
intermediate.	2	6

The last layer (fourth), consisting of magnesite powder, was spread with the fettling machine over the whole surface of the bath. 26 tons of powder was used for this layer, i.e., it replaced 11 layers applied by ordinary sintering methods. With the rapid sintering method, less magnesite powder was used than on previous occasions so that the sintered part of the bottom was somewhat thinner. However, subsequent operation of the furnace showed that the change in the thickness of the furnace bottom had no effect on its service life.

The fourth layer was heated for 5 hours 10 minutes after which time it was slagged with scale alone which was spread over the bottom, banks and walls. 30,000 kg of scale was used. A well-heated layer of magnesite powder quickly absorbed the scale which has a lower melting point than slag. The slagging of the bottom took 3 hr 17 min .

It should be mentioned that with the rapid method of bottom sintering, a slightly larger quantity of fluxing additions was used than in previous sintering operations.

The temperature of the bottom during the sintering was raised at the rate of 5°C per hour from 510 to 710°C. The greatest rise in the temperature of the bottom took place before the charging of the materials for the second layer. The temperature of the main roof during the sintering was 1550-1600°C. The highest temperature of the roof—1700-1730°C—was maintained during the slagging of the brickwork and sintering of the first layer. The temperature at the top of the checkers fluctuated between 1050 and 1300°C, and that of the gas checkers between 950 and 1200°C.

The consumption of the blast-furnace gas remained constant during the sintering, and the consumption of coke oven gas varied between 4800 and 6500 nm^3/hr. The highest rate of coke oven gas firing was during the slagging of the brickwork and the heating of the fourth layer. During the sintering, the air was forced into the working space of the furnace by an air fan.

Samples were taken from the first three layers with the use of steel tubes with flattened ends. The quality of the sintered layer was estimated from the appearance and the type of fracture, and then the samples were passed on for chemical analysis and for the determination of specific weight.

The samples from the bottom were dense, dark brown in color, well sintered, and without noticeable boundaries between separate layers; the fracture took place along the grain boundaries. The specific weight of the samples varied between 2.78 and 3.65 g/cm^3, depending on the position along the furnace from which they were taken and depending on the layer. The average specific weight of the sintered layer was 3.26 g/cm^3

Subsequently, the increase in the density of the bottom during the sintering was determined in the laboratory. It was established that the bottom becomes substantially denser even during a short period of heating. It must be stressed that the density of the bottom varies along the furnace and from layer to layer. The porosity varies between 3.8 and 33% and this is to be expected since the porosity depends on various factors: the distribution of fluxing materials, the method of charging the materials, the distribution of the materials over the bottom as well as on the method of taking samples all of which (except one) contained considerable quantities (65-87%) of MgO, i.e., they were sufficiently heat resistant. It was noted that samples taken from the same layer but at different positions along the bottom varied in chemical composition. Thus, for instance, the samples of the second layer taken near the first, second and third charging doors contained from 62 to 69% MgO, and the samples taken near the fourth and fifth doors contained from 73 to 83% MgO. This phenomenon can apparently be explained by inadequate preparation of the charging materials. During the petrographic investigation of polished samples it was established that the main components are periclase (MgO) and magnesioferrite ($MgO-Fe_2O_3$). Their quantity constitutes approximately 85%. The melting of periclase grains is insignificant. The periclase and ferrite grains are cemented with interlays of silicates which constitute approximately 10-15%.

The first heat was carried out with the use of liquid pig iron, which was not previously the practice at the Works. The weight of the charge constituted approximately 92% of the nominal furnace capacity, and the duration of the heat was 10 hours 10 minutes. After 42 heats a sample of the bottom was taken from near the second charging door. According to its appearance it could be divided into two samples whose analysis is given in Table 3.

The first preventive repairs to the bottom were carried out after 55 heats and took 2 hours 20 minutes.

The wear of the bottom was uniform.

TABLE 3. Chemical Composition of Samples Taken during the Operation of the Furnace, %.

Sample No.	SiO_2	Al_2O_3	CaO	MgO	Fe	FeO	Fe_2O_3	MnO	P_2O_5	Cr_2O_3
1	7.08	4.2	18.62	31.82	25.36	25.84	7.54	3.33	0.82	0.07
2	9.72	5.16	22.46	45.51	11.02	13.12	1.18	1.86	0.82	Сл.

TABLE 4. Service Life of the Bottom and Idling Periods of the Furnace

Item	Month			
	I	II	III	IV
Idling periods of the furnace during repairs of the bottom, %	0.31	1.10	0.65	0.75
Service life of the bottom (in numbers of heats)	77	22	35	36

By sintering the bottoms of basic large-capacity open-hearth furnaces in four layers with the use of scale for slagging*, we found it possible to reduce the time of sintering by more than one third.

Idling periods of the furnace during the repairs to the bottom were reduced to 0.7% and the service life of the bottom between consecutive repairs increased to 40.9 heats (Table 4).

Extensive use of the rapid sintering technique for large capacity furnaces will make it possible to reduce the idling periods of furnaces substantially and to increase the output of steel by thousands of tons.

*This method was first proposed by M.M. Privalov, Senior Foreman at the Kuznetsk Metallurgical Combine (Editor's note).

THE 250 LIGHT-SECTION ROLLING MILL

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Krivoi Rog Metallurgical Works

The 250-1 continuous medium-section rolling mill at our Works is used for the production of round sections of 10-20 mm diameter, periodic sections Nos. 10-20 for reinforced concrete, 16 mm square section, and equal angles 35 × 35 × 4-5 mm and 40 × 40 × 4-5 mm.

The starting material for the 10 mm round and No. 10 periodic section is a square of 12 mm side and 12 m long. The other sections are rolled from 80 mm square billet.

The billets are placed by cranes onto two loading grates and then transferred to rolling tables which take them to the reheating furnace. The billets are charged into the furnace by means of a pusher mounted at the charging side door of the furnace. The billets are heated in a two-zone continuous furnace with a wide solid bottom inclined at an angle of 8°15' to the horizontal. The furnace is fired with a mixture of blast-furnace and coke-oven gases of 1400 kcal/nm^3 calorific value. There are 28 injection-type burners, half of them in the heating zone.

Gas and air are heated in a metallic recuperator and a ceramic one; gas is heated to 200-250°C and air to 500-600°C. The furnace can accommodate 186 80 × 80 mm or 248 60 × 60 mm billets at the same time. The temperature of the steel delivered to the mill depends on the steel grade and the operating procedure of the mill and varies between 1140 and 1180°C.

The billets are moved through the furnace by a lever pusher and the hot billets are delivered from the furnace through a side door by means of a discharge pusher.

The mill consists of 23 working stands, constituting one roughing and two parallel finishing groups. In the roughing group of the mill, two sections are rolled in parallel and therefore a distributing-and-discharging machine which diverts the billets into the appropriate pass of the first stand was installed between the furnace and the first stand.

Shears of 280 tons cutting force were installed in front of the first stand for cutting the billets into several lengths depending on the section rolled, and also for cutting off the front end or for emergency cutting.

The roughing group of the mill consists of 7 working stands equipped with individual drives with a wide range of rolling speeds.

After passing through the roughing group, the metal is divided into two streams and the rolling process is carried out in two finishing groups, one piece in each group. Rotary flying shears for cutting off the front end of the piece are provided for each stream. In the case of a breakdown, the flying shears are used for cutting any section which becomes jammed. At the same time the cracker shear installed in front of the right finishing group is put into action.

Each finishing group of the mill consists of eight stands with alternate vertical and horizontal rolls.

A particular design feature of the vertical stands is the fact that they consist of two parts; first a massive large base supporting an electric motor and a mechanism for mounting the vertical working rolls, and second a box containing the housing, bearing chocks and working rolls.

Each stand of the finishing group has a separate drive. The maximum speed of the last finishing stand depends on the section rolled and varies between 7 and 14 m/sec.

After passing through the finishing group, the rolled piece is transferred to a cooler or to coilers.

The finished product is cooled in a two-side cooling bed which is 125 m long and is provided with packing machinery and discharge rolling tables. The rolling pieces are cut to the required length with 600-ton shears mounted behind the cooler.

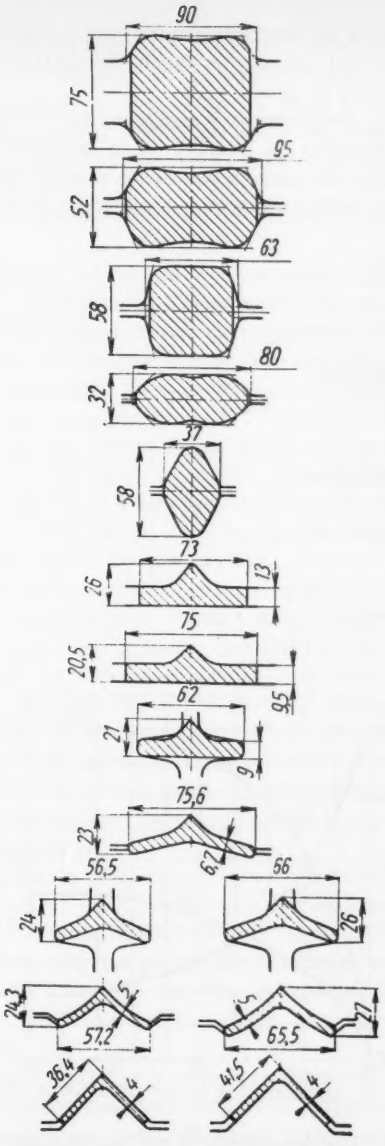
According to the design, rectangular and round sections can be supplied either in coils or bundles. For this purpose the mill is equipped with four garret coilers designed for a coiling speed of 5.5-15 m/sec. There are four strip coilers for coiling flat sections at a speed of 6-11 m/sec.

Bundles and coils are bound by binding machines and are transported on a plate conveyor to a hook conveyor, and then to the coil lifters which transfer them to the packaging section.

The technique of rolling on the 250 continuous mill is distinquished by the fact that the rolled piece passes simultaneously through all the working stands. In this process it is very important that the speed of rolling is very strictly controlled and this can be achieved by the automatic control of the number of revolutions of the rolls.

The speed of the motors in the range from zero to the basic speed is controlled by variations in the voltage on the rotor.

The control of the speed above the basic speed is effected by the combination method, i.e., by the voltage at the rotor and by its excitation. The combined control ensures a rapid response of the system. During the operation of the mill the speed control system maintains



Pass arrangement for rolling an equal angle.

the number of revolutions very accurately. During the entry of the piece into the stand, the dynamic drop of the speed does not exceed 1-2% and the static 0.5%; the magnitude of this speed drop can be controlled by means of a rigid compounding system, this control being especially important during the rolling of shaped sections since with a soft characteristic of the motor a better grip of the rolled piece by the rolls is achieved. The motors of the auxiliary equipment (shears, coilers, cooler, etc.) vary the speed according to the speed of rolling. The master variable for these mechanisms in the electronic regulator is the voltage of the speed-voltage generator of one of the rolling stands.

In the roll design for this mill the specific features of the mill were taken into account.

Box passes are used in the first three stands of the roughing group for rolling all sections except 10 mm rounds and No. 10 section. The pass arrangement in the remaining stands of this group is based on the oval and oval-on-edge system. For rolling these sections, however, a box pass is only in the first stand of the roughing group which always serves as a scale breaker and has a small draft coefficient (1.08-1.13).

The passes of the first stand are distinguished from the remaining box-type passes by shallow grooves in the rolls and large distance between the rolls.

The oval and oval-on-edge pass arrangement is used in the finishing groups of the mill for rolling round sections. The same system is also applied to the production of square steel sections in the first stands of the finishing groups. The pass arrangement of the mill is standardized for rolling all round sections in the same passes. For instance, when 18-20 mm round sections are rolled, the rolls with a 15 × 42 mm oval pass are installed in the 10th stand, and when 12 - 16 mm round steel sections are rolled the same oval pass should be in the 8th stand. In addition, combination rolls (with various grooves) are extensively utilized at the mill. In this way it is possible to roll several sections with the same rolls without roll changes. This pass arrangement and roll mounting has made it possible to simplify the storage of rolls and reduce the stock of rolls substantially.

The oval and oval-on-edge pass arrangement is at present extensively used in light-section continuous mills with the appropriate alternation of vertical and horizontal rolls. In this way it is possible to eliminate the operation of turning the piece between the stands and thus to simplify the auxiliary equipment of the rolls substantially. In addition, this system of pass arrangement involves only small changes in the shape of the rolled piece in each pass, thus providing a better final product.

Until recently angle sections were rolled only on mills arranged in train since it was difficult to roll them at high speed on continuous mills because of the lack of automatic control of stretching and looping. Nevertheless, a high temperature of the rolled piece on all stands, a better design of the working stands, the use of roller bearings for the rolls, etc., provide a practical possibility for rolling shaped sections on continuous mills.

A successful production of angles (see Figure) was carried out on the 250 mill where the passes in the rolls of the first 4 stands of the roughing group of the mill were left the same as for rolling circular sections. The next five passes (two in the roughing group and three in the finishing) are common for rolling 35 and 40 mm equal angles. Two out of the six shaped passes in the finishing group of the mill are in vertical stands in which the rolls serve for controlling the width of the flanges and the formation of the side edges.

It was found in practice that these sections are rolled best at a speed of 6.5-7.5 m/sec. Any further increase in the speed of rolling does not result in an increased output of the mill because irregularities occur more frequently, the quality of the rolled section deteriorates and the time used for mill adjustment increases. This is explained by the lack of equipment for controlling the size of the loop between the working stands and by the fact that the operator cannot change the regime of rolling at high speeds.

The development of equipment for determining and controlling the behavior of rolled pieces between the stands will make it possible to simplify the adjustment of continuous mills and to increase their efficiency.

NONOXIDIZING HEATING IN THE ROLLING-MILL AND FORGE-SHOP FURNACES

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During the heating of steels in furnaces, scale is formed under the action of oxygen and other oxidizing gases present in the furnace atmosphere. The loss of metal in the rolling-mill and forge-shop furnaces amounts to 2-3%, and the cost of the direct loss of metal is equal to the cost of the total fuel used for heating. In addition the formation of the scale makes the operation of the furnace and the use of other equipment difficult; it frequently results in defective product during rolling and forging, causes considerable losses in connection with the removal of the scale, etc. Therefore, the problem of nonoxidizing heating is very important from the point of view of the national economy.

The composition of the furnace atmosphere containing oxidizing gases (CO_2,H_2O, O_2 and SO_2), neutral gases (N_2) and reducing gases (CO,H_2) is of critical importance for the process of metal oxidation. The content of the oxidizing gases is high under ordinary conditions of fuel combustion and hence the atmosphere in the furnace has oxidizing properties.

If the fuel is burnt with much less than the theoretical amount of air required, then, simultaneously with the reduction in the content of the oxidizing gases there will be an increase in the quantity of reducing gases, and under certain conditions the furnace atmosphere may become neutral and ensure a nonoxidizing heating of the metal.

Investigations have shown that for heating steel without scale formation in the rolling-mill and forge-shop furnaces the quantity of air for combustion should be half of the theoretical quantity, that is, the coefficient of air consumption, α, should be equal to 0.5.

To obtain a high temperature in the working space of the furnace under these conditions it is necessary to use fuel of a high calorific value and burn it with heated air or oxygen.

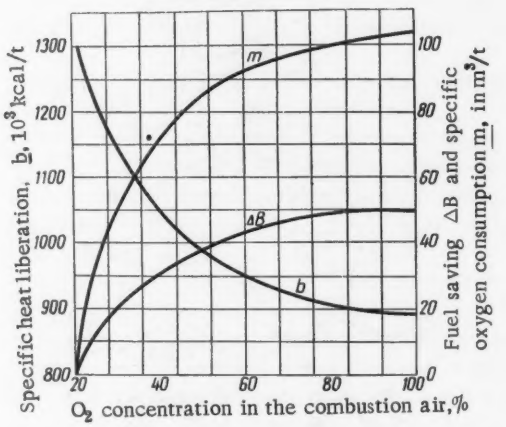

Fig. 1. The dependence of the specific heat liberation, fuel saving, and specific oxygen consumption, on the oxygen concentration in the combustion air during the burning of natural gas at $\alpha = 0.5$ and with the afterburning of the products of incomplete combustion before the regenerators.

Calculations show that during the combustion of the natural gas in the rolling-mill and forge-shop furnaces the temperature of the combustion air should be at least 700-800°C, and during the combustion of the coke-oven gas it should be approximately 500°C. If the combustion air is enriched with oxygen, this temperature is reduced.

The second important problem which arises in connection with the nonoxidizing heating of steel is the utilization of the heat available in gases after the complete chemical combustion of the fuel, which heat constitutes 55-65% of the calorific value of the fuel when $\alpha = 0.5$.

There are two main types of furnaces for utilizing this heat.

In furnaces of the first type the products of the incomplete combustions are burnt outside the working space of the furnace—before the regenerators or recuperators—and the heat is used for warming up fresh air for combustion. Thus, for instance, in chamber furnaces at a temperature of 1300°C, less than 20% of the physical and chemical heat of the waste gases is used for heating the air to 1000°C. Therefore these furnaces operate at a high specific heat liberation which exceeds by three or four times the specific heat liberation under normal firing conditions.

The fuel consumption in these furnaces can be reduced by using oxygen. Figure 1 shows that the use of oxygen results, on the whole, in a small saving of fuel since with the reduction in physical heat losses owing to the smaller quantity of flue gases one also reduces the quantity of heat introduced with heated-up air owing to the reduced content of nitrogen. At the present cost of oxygen, expenditure connected with the use of oxygen considerably exceeds the savings gained by the reduced fuel consumption.

In furnaces of the second type the afterburning of the products of the incomplete combustion takes place in that part of the working space of the furnace which is separated from the heating chamber by an intermediate roof, or in special tubes made of heat-resistant materials and located in the working chamber of the furnace.

By this method it is possible to utilize the heat of the incomplete chemical combustion of the fuel directly for heating the metal. However, with an intermediate carborundum roof 50 mm thick, the coefficient of fuel utilization increases by only a factor of 1.5 and at the same time the specific heat liberation remains very high. In addition, such an intermediate roof makes the design of the furnace difficult and reduces its service life.

Thus, there is an excessive fuel consumption in both types of furnaces so that their economic effectiveness is substantially reduced.

It is known that the loss of metal in burning increases rapidly with the increase in temperature. Therefore the heating process can be conveniently divided into two stages; the first—heating to the temperature at which the metal is only slightly oxidized, and the second—final heating of the metal at the coefficient of air consumption $\alpha = 0.5$.

The heat of the incomplete chemical combustion of fuel is then utilized for the preheating of metal when its temperature is not yet high enough for significant oxidation.

An experimental heating of specimens in the continuous furnace for the determination of the permissible temperature of the metal during preheat was carried out at the Dzerzhinskii Works. A rapid increase of metal loss begins at temperatures above 950-1000°C. Therefore it can be assumed that the first stage of heating in the atmosphere of products of incomplete combustion can be carried out up to 900-950°C without any significant oxidation of metal. These results were also confirmed by special investigations carried out with the use of an experimental chamber.

Two-stage nonoxidizing heating can be employed in chamber as well as continuous furnaces.

Figure 2 shows a chamber furnace for two-stage nonoxidizing heating. The furnace operates as follows. After the completion of the heating of the metal in one of the chambers (for instance in chamber I) the metal is discharged and cold metal is charged into the furnace. By that time the charge in chamber II is heated to 900-950°C. Burners in chamber I are turned on and gas with air considerably below the theoretical requirements ($\alpha = 0.5$) is supplied. The products of incomplete combustion enter chamber I (the flue damper is shut) where they burn with secondary air introduced through the

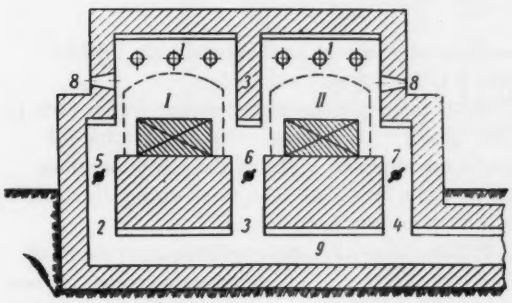

Fig. 2. A chamber furnace: 1) Tuyeres; 2)3)4) flue gas ducts; 5)6)7) dampers; 8) burners; 9) flue.

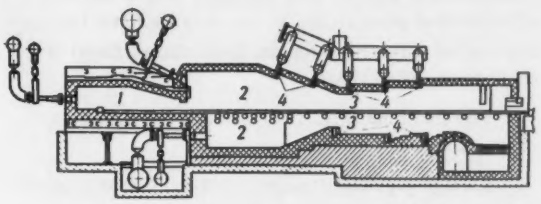

Fig. 3. A continuous furnace; 1) Soaking zone; 2) heating zone; 3) preheating zone; 4) tuyeres.

tuyeres. The flue gases are taken by flue gas ducts to the recuperator.

By the time the metal in chamber I is heated to 900-950°C the heating in chamber II is completed, the burners are turned off, the hot metal is discharged, the cold metal is charged and the burners in chamber I are turned on. At the same time the damper in chamber I is closed and the damper in chamber II is opened. Then the second stage of heating takes place in chamber I with less than the theoretical amount of air, and the first stage of heating takes place in chamber II where the products of incomplete combustion leaving the chamber I are burnt.

The intermediate gas duct serves for the partial removal of the products of incomplete combustion past the heating chamber to heat the metal in this chamber to a maximum temperature of 900-950°C. The flue gas flow is controlled by the gas damper.

One can control the temperature in the preheat chamber by regulating the incomplete combustion of gases which enter the chamber instead of by the intermediate passage of theflue gas.

Calculations show that in the furnaces described above one can utilize almost fully the heat of the incomplete chemical combustion and hence the heat liberation in those furnace is not higher than in the furnaces in which normal fuel combustion is employed.

Figure 3 shows a diagram of a continuous furnace for the two-stage nonoxidizing heating of metal.

The fuel is burnt at air consumption coefficient $\alpha = 0.5$ in the soaking and heating zones of the furnace. In the preheating zone the products of incomplete combustion burn with the secondary air introduced through the tuyeres. The metal in the preheating zone is heated to 900-950°C just as in the preheating chamber of a chamber furnace. In this way 85 to 90% of the heat of the incomplete combustion is utilized.

Analysis of this investigation shows that the heat losses in the heating zone in these furnaces have a great effect on the specific heat liberation. Thus, if air is heated to 550°C and heat losses in the heating zone increase from 5 to 10% the specific heat liberation is more than doubled.

Since the main source of heat losses in the heating zone are water-cooled hearth tubes, the furnaces should be designed with a solid hearth in the heating zone.

The existing continuous furnaces can be converted to nonoxidizing ones in two stages. A project for such a modification of the furnace of the 550 mill at the Dzerzhinskii Works has been developed by the Stalproekt. Calculations show that by heating the metal in the preheating zone to 950°C, one can utilize approximately 85% of the heat of the incomplete combustion of fuel. The specific heat liberation will constitute 670 kcal/kg.

The above results show that it is possible to carry out the nonoxidizing heating of metal in the rolling-mill and forge-shop furnaces while maintaining a high efficiency of fuel utilization in those furnaces.

HYDRAULIC REMOVAL OF SCALE AT PLATE MILLS

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To improve the quality of the rolled product, to facilitate the work of mill operators and to increase the efficiency of plate mills, at our shop we have installed and started a machine for the hydraulic removal of scale from rolled sheets.

The machine (Fig. 1) consists of the following main parts:

1) High pressure storage (receiver) comprising six cylinders of 1150 liter capacity each.

2) An electropneumatic contact control arrangement consisting of the receiver control (relief valve), electropneumatic control, three circulation valves and the distribution equipment.

3) A water tank of 2 m^3 volume with quartz filters, 2 m in diameter and 2.75 m high.

4) Six three-plunger high-pressure pumps each 15 m^3/hr capacity. The pumps serve for charging the water to the receivers. Each of the pumps has three plungers and a crankshaft with three throws displaced by 120° with respect to each other.

5) Two three-stage high-pressure compressors of 16.6 m^3/hr capacity. Each compressor consists of three interconnected cylinders located on a common vertical axis. The three pistons are actuated by a single connecting rod in vertical directions. The compression chambers are connected by helical tubing placed in the water tank on the compressor body.

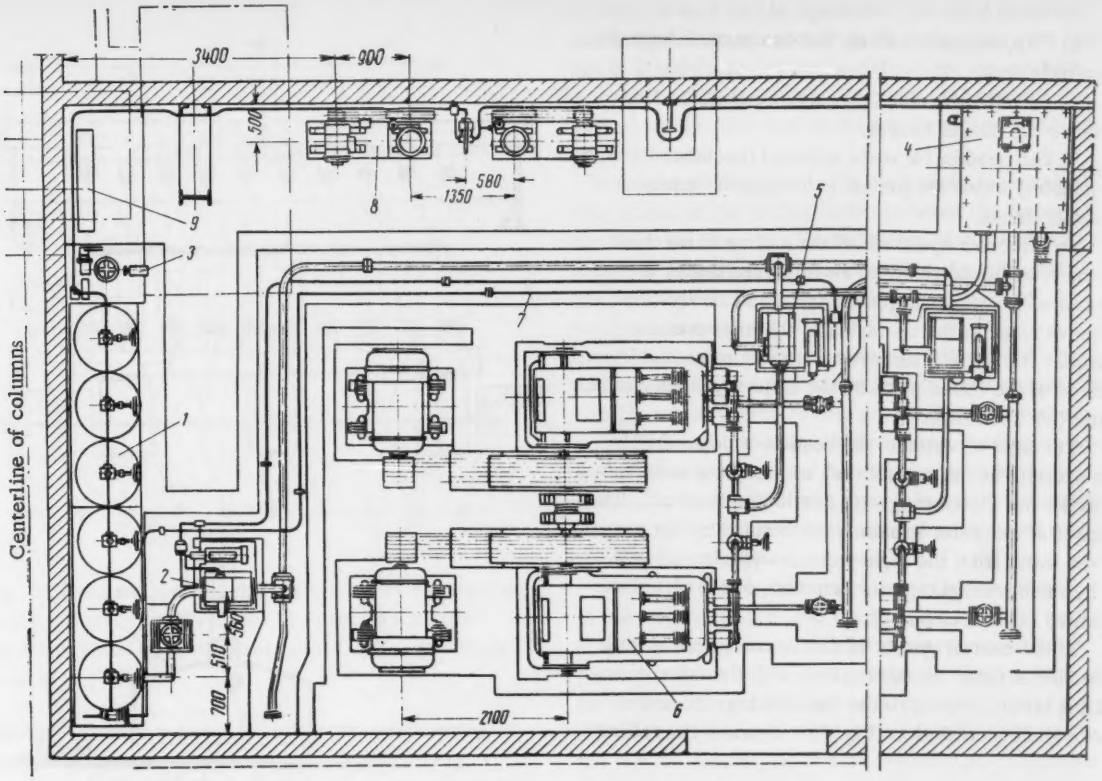

Fig. 1. Machine for hydraulic descaling (part of the system); 1) Cylinder bank 2) receiver control; 3) electro-pneumatic control; 4) water tank; 5) 6) valves; 7) pipes; 8) compressors; 9) distributor.

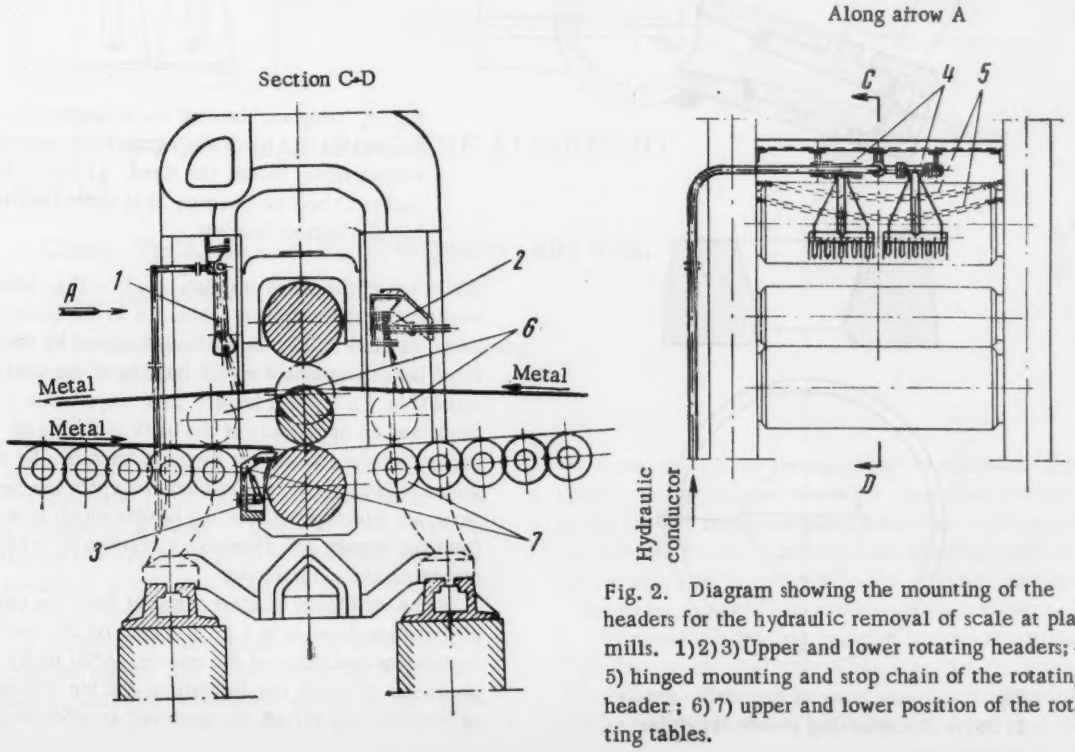

Fig. 2. Diagram showing the mounting of the headers for the hydraulic removal of scale at plate mills. 1) 2) 3) Upper and lower rotating headers; 4) 5) hinged mounting and stop chain of the rotating header; 6) 7) upper and lower position of the rotating tables.

6) Two magnetic valves, button operated from the operator's desk.

7) Two distributors (with valve control) feeding water to the nozzle header.

8) Ten headers for scale removal (each header with 10 nozzles) which are placed in front and at the back of the mill stands.

An effective operation of the hydraulic machine has been achieved owing to an improved design of several elements, a simplified system of control and improvements in the operation of the high-pressure pumps. The scale is removed from the upper and lower surface of the rolled plate by the impact of high-pressure water (90-100 atm).

The feed of water to the headers is controlled by the operator at the control desk of the plate mill who switches the electromagnetic distributor on or off. The impact of the water stream is maintained by the pressure of the water from the high-pressure cylinders as well as by subsequent automatic switching on (or off) of the required number of pumps.

In the control system of distributors at plate mills provision is made for interlinking with the drive of the rolling tables. Owing to the interlinking, the water enters the upper headers (Fig. 2) only when the rolling

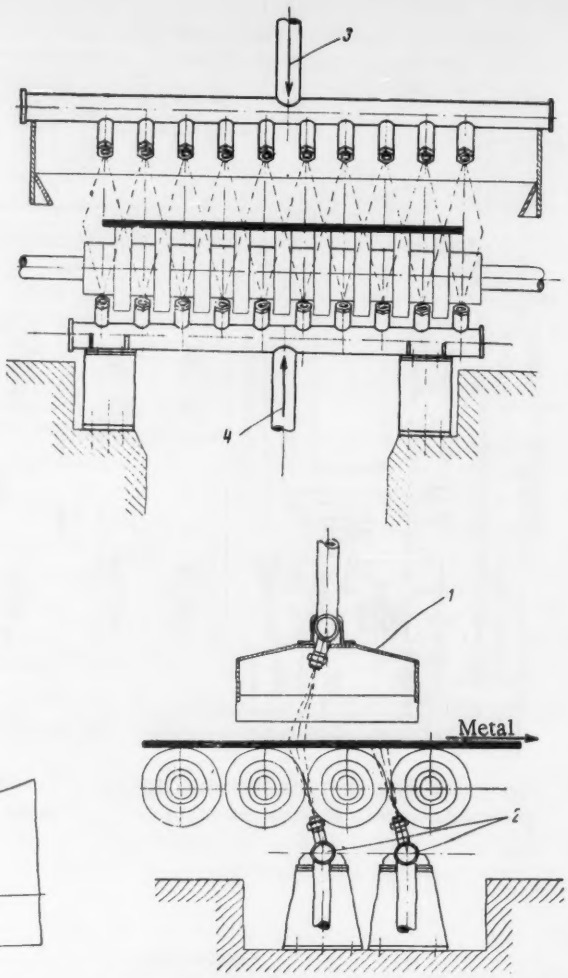

Fig. 4. Diagram showing the arrangement of the headers for the hydraulic removal of scale on the rolling tables before the stand. 1) Protective jacket; 2) bottom headers; 3) 4) water feed to the top and bottom headers.

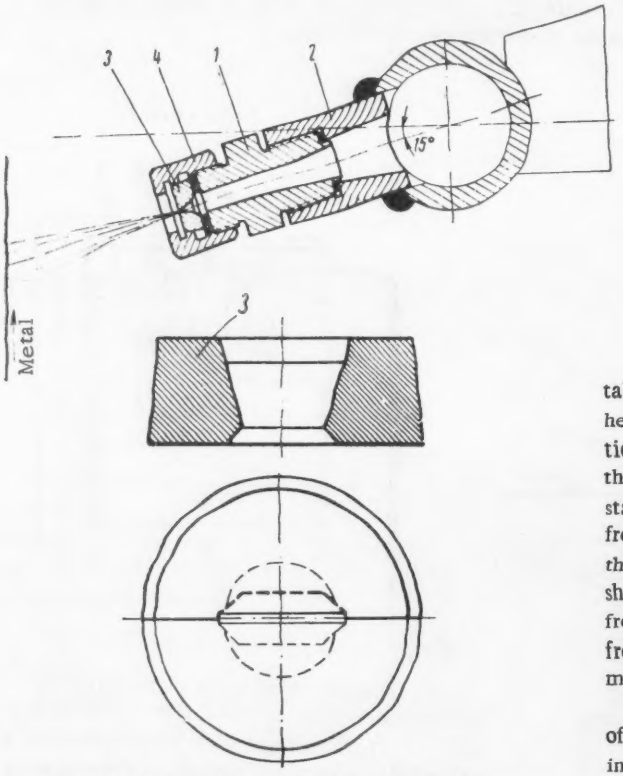

Fig. 3. General view of the header nozzle. 1) Body; 2) connecting piece; 3) nipple; 4) union nut.

tables are in the upper position, and to the lower headers—when the rolling tables are in the lower position. At plate mills, the scale is removed by means of three headers mounted on the housing of the working stand (Fig. 2); a lower header and upper one at the front, and an upper only at the back of the stand. At the same time, to prevent frequent fractures, by the sheets rolled, of the rigidly mounted upper header at the front, we developed a rotating header which is protected from the impact and excessive vibrations by a hinged mounting and a chain stop.

The efficiency of scale removal from the surface of the plate depends to a great extent on the positioning, inclination and shape of the nozzle. After many experiments in which the dimensions and the inclination of nozzles were varied, we achieved an effective uti-

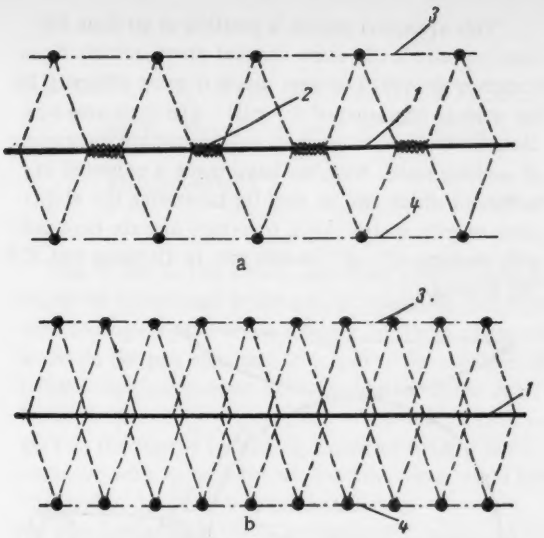

Fig. 5. Diagram showing the positioning of the nozzles: a) old system; b) new system; 1) sheet; 2) scale not removed; 3)and 4) centerlines of the upper and lower headers respectively.

lization of water pressure, a minimum consumption of water, and a most efficient removal of scale (Fig. 3).

During the operation of the hydraulic descaler, the thread on the nozzle broke down frequently and the nipples were forced out of the union nuts thus resulting in some hazard for the operators. We have modified the design of the nozzles (Fig. 3) so that they now consist of a body screwed into a connecting piece which is welded to the header; a union nut (4) has a conical part and a shoulder so that the nipple can be retained inside the nut.

At one of the plate mills the hydraulic descaler was placed at the rolling tables between the furnace and the mill (Fig. 4). Heated billets (for instance 25 × × 1400 × 1200 mm), whiledelivered to the mill, pass through a special hydraulic wash chamber where the scale is removed.

The chamber consists of three headers (one upper and two lower) and 30 nozzles, ten in each header. The nozzles are staggered so that a complete removal of scale from the billet can be obtained. Previously, when we had two headers – one lower and one upper – with nozzles spaced at 150 mm, the scale was not completely removed (Fig. 5a) and some areas of unremoved scale remained on the plate.

Owing to the selection of the optimum dimensions of the nozzle orifices and the positioning of the nozzles (Fig. 5b), the quantity of defective and second grade sheets has been substantially reduced; whereas prior to the introduction of the hydraulic descaler more than 4 % of sheets were rejected during sorting because of the surface defects, now there is hardly any defective product .

In addition, the introduction of the hydraulic descaler made the work of the operating personnel easier.

APPARATUS FOR MEASURING THE ALIGNMENT OF ROLLS

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Scientific Research and Design Institute of Mining and Ore-Beneficiation Machinery

Several steel rolling mills in the USSR use two-, four-, and six-roll mills. A normal operation of these mills depends to a large extent on the accuracy of aligning the working rolls in one vertical plane. If this condition is not adhered to, the rolls are displaced axially during the process of steel deformation thus causing disruption in the normal operation of the mill, reducing the service life of the supporting rings and bearings as well as impairing the quality of the rolled steel.

Some mills have special adjusting equipment (for instance wedge supports under the screwdown mechanism on the six-roll mills) for positioning the working rolls in one vertical plane; on several mills the alignment of the working rolls is achieved by the accurate manufacture of the bearing chocks and windows of the housing.

The working rolls are mounted in one vertical plane depending on the character of the deformation of the rolled piece. This method involves several difficulties and frequently requires a long time for completing the

operation. The usual measuring instruments cannot be usefully employed for the adjustment of the mill because there is no base for mounting them and because of the convex shape of the working roll barrel.

For determining the alignment, for facilitating the adjustment of the mill and for a rapid and accurate mounting of the working rolls in one vertical plane we designed, made, and put into operation at one of our Works a special very simple apparatus (Fig. 1). One of the working rolls, to which two end guide blocks of the apparatus are attached at the ends of the roll barrel, serves as a base for the measurements. By moving the slider with the indicator along the rod from one extreme position to another we can measure the actual displacement of one working roll with respect to another. The magnitude of the misalignment of the rolls is determined from the difference between two readings of the indicator.

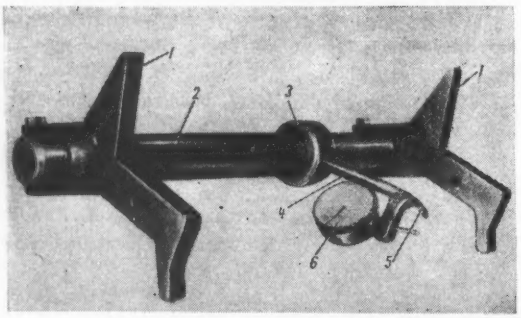

Fig. 1. Apparatus for measuring the alignment of working rolls of a six-roll mill. 1) End guide blocks; 2) rod; 3) slider; 4) handle; 5) feeler; 6) indicator.

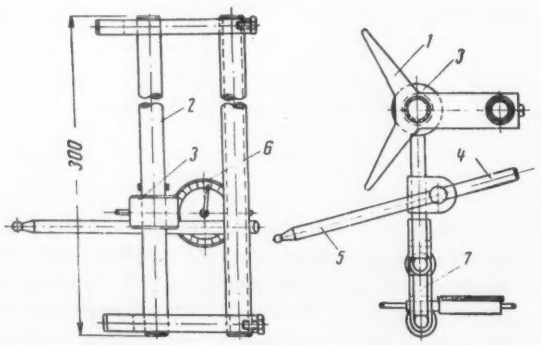

Fig. 2. A universal attachment for measuring the alignment of the working rolls of two-high, four-high and six-high mills. 1) End guide blocks; 2) rod; 3) slider; 4) handle; 5) feeler; 6) indicator; 7) hinged arm.

This apparatus makes it possible to position the working rolls in the same vertical plane quickly to an accuracy of 0.02-0.08 mm which is quite adequate for the normal operation of the mill. The apparatus was designed for a six-high mill and for certain dimensions of working rolls. Now, we have made a universal attachment which can be used for measuring the alignment of rolls on two-high, four-high and six-high mills with working rolls of 150-600 mm in diameter and 325-700 m long.

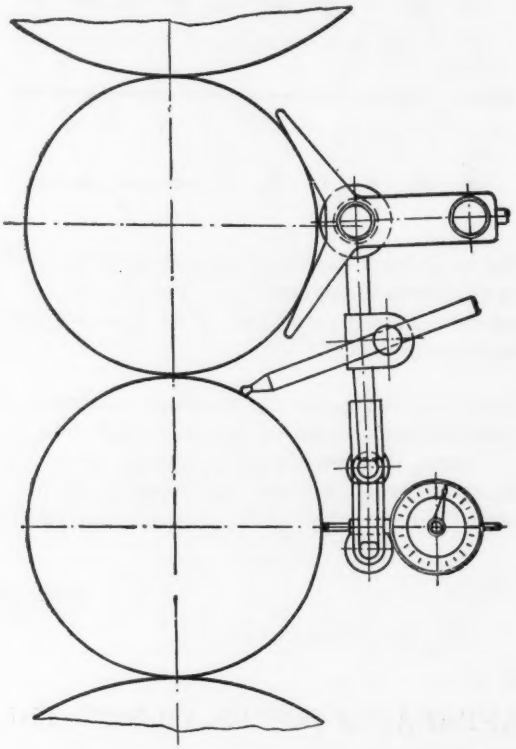

Fig. 3. Diagram showing the mounting of the universal attachment.

The universal attachment shown in Figs. 2 and 3 differs from the attachment described above by the extension pieces to the handle and the hinged arm with the indicator so that the indicator can be mounted exactly parallel to the centerline on rolls of various diameter (Fig. 3). In addition the feeler can be moved, set at different angles, and fixed in the required position. Another advantage of the universal attachment is that it can be used to measure the convexity of the working roll directly on the mill.

The attachment can be made either universally suitable for several mills or it can be made specially for one particular mill.

In both cases these simple attachments can be made locally at works and can be used for the positioning of rolls during installation, roll changing and normal operation of the mill.

REPAIRS TO THE HEATING FACILITIES OF ROLLING MILLS

M.E. Portnoi and V.I. Zhivago

VNIIOChERMET

One of the factors which contribute to an increased output of rolled steel is the correct operation and timely and thorough repairs to the heating facilities. However, works do not pay adequate attention to the problem of furnace repairs. In spite of the resolution of the All-Union Conference of Rolling Mill Operators in 1955 regarding the repairs to heating facilities strictly in accordance with an established schedule, some works have no schedule at all or it is not adhered to.

For instance, at the "Zaporozhstal'" Works the repairs of one of the groups of the slabbing-mill soaking pits were due in February 1958 according to the schedule. The repairs were postponed until May, then until Sept., but even then no repairs were carried out. Delayed repairs result in an excessive wear of not only the refractory brickwork but also of parts of the valve changing mechanisms and cover lifting equipment as well as of the steel structure of the soaking pits.

The heating facilities which are now in use at metallurgical works in the USSR vary greatly in design, type of fuel and type of steel heated; therefore, there is no single classification of repairs and no standard timetable has been established. We tried to establish such a classification on the basis of the study of practice with respect to heating facilities of most common types at leading metallurgical works; regenerative, large capacity and recuperative furnaces and continuous furnaces (two- and three- zone) were considered.

The maintenance of heating facilities are divided into hot, medium cold, extensive cold and major overhauls. Approximate timings and standard programs of repairs of heating facilities are shown in the table. If these standards and the classification of repairs are accepted, it is possible to plan the repairs of heating facilities for a long time in advance (quarterly or annually).

Before the heating unit is shut down for repairs its technical condition is inspected and then the extent, type and any particular detail of the repairs required for separate parts of the furnace are determined. From the results of the inspection, a repair schedule which includes the number of workers from each trade, time limits and other details, are determined.

A rational preparation of the repairs to heating facilities must include a well-thought-out program of repairs which includes the following steps:

a) procurement of the necessary materials (bricks, refractory powders, tools, etc.) and the assembly of those materials near the furnace to be repaired;

b) preparation of machinery (conveyors, single-rope hoists), assembly and adjustment of machinery so that all the equipment is ready when the repair operations begin;

c) dismantling the maintenance doors in the recuperators, regenerators and slag pockets;

d) disconnecting the supply of gas (or fuel oil), cooling water and air to the unit;

e) cooling separate parts of the heating furnace (if possible), water spraying by means of hoses is applied;

f) preparation of the necessary equipment, auxiliary materials and tools (boxes, staging, box caissons),

g) installation of the necessary ventilation and arrangements for rest places;

h) preparation of planking and casing materials.

We describe below the preparation and carrying out of repairs to heating facilities on the basis of the practice of metallurgical works in the USSR and taking into account the mechanization of individual repair operations. Carrying out the repairs by this method made it possible to reduce the duration of the repairs and increase the length of the campaign of heating facilities.

REPAIRS OF RECUPERATIVE SOAKING PITS.

After the brickwork has cooled down and the walls of the cell have been dismantled, the rubble is removed

Approximate Timings and Durations of Repairs to Heating Facilities

Type of repairs	Recuperative soaking pits		Regenerative soaking pits		Continuous furnaces	
	Interrepair period	Duration of repairs	Interrepair period	Duration of repairs	Interrepair period	Duration of repairs
Hot	—	3—5	—	3—5	—	7
Medium cold	6 months	25—35	3 months	25—35	6 months	25—35
Extensive cold	2 years	80—90	2 years	100—110	2 years	110—120
Major overhaul	10 years	180—200	10 years	220—240	10 years	140—160

and the work on breaking up the slag on the bottom, and breaking up the bottom itself including the ports and hot air duct, is carried out. At the same time the roofs of the recuperators are demolished, the checkers are broken down and the remaining walls of the recuperators and underchecker installations are dismantled.

When the bricklaying begins, first the underchecker installations, then the walls of the recuperator and the checkers, for which special shaped refractory bricks are used, are laid. Then the recuperator roofs which are insulated with a layer of foam firebrick laid flat are erected on the planking.

At the same time, separate parts of the furnace working space are built. The lower half of the hot air duct is lined with a layer of foam firebricks. The planking for the upper part, which is laid concurrently with the base of the burner, is erected. The brickwork of the burner throat is covered with a layer of chromite ore. At the same time, the walls of the cell are laid at first with chrome-magnesite bricks (12-15 flat-laid rows) and then with silica brick or firebrick.

For a tight fitting of the lid a sand seal, made of special shaped stones or standard bricks, is built along the perimeter of the cell.

The dismantling and erection operations are carried out simultaneously with the bricklaying.

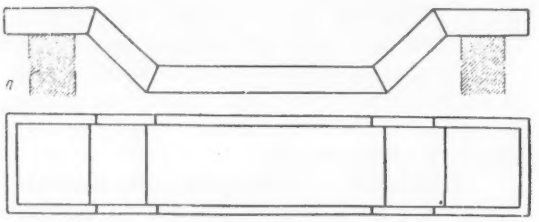

Fig. 1. Diagram of the staging for furnace repairs: a) Wall of the cell.

The use of staging of the type employed at the "Azovstal'" Works (Fig. 1) during the hot repairs makes the work of the bricklayers much easier. At the Zakavkaz Metallurgical Works a staging (Fig. 2) with a special caisson for the repair to the burner throat has been designed.

REPAIRS TO REGENERATIVE SOAKING PITS

First of all the walls of the cells and the roofs of the regenerators are dismantled, then the slag is removed from the bottom, and the bottom and the separating walls are demolished. The checkers of the regenerators are pulled down simultaneously on both sides in four cells.

The bricklaying is carried out in the reverse order. After the walls of the regenerator are laid, the roof planking is erected and then the roofs are laid and insulated with foam firebricks. Adjoining the roofs of the

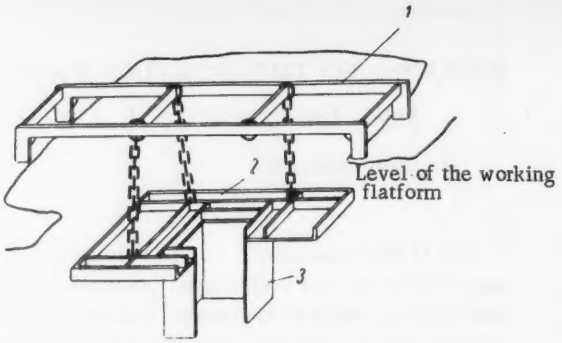

Fig. 2. Staging for furnace repairs: 1) Frame; 2) platform for hot repairs; 3) caisson.

regenerator are the side arches which are laid with 300 mm bricks in two rows. The support of the arches is common with the walls.

During the repairs, the roof-lifting equipment and separate parts of the steel structure are first dismantled and then assembled, and the covering structure is checked. By using the equipment for the transfer of roofs at the Dzerzhinskii and "Zaporozhstal'" Works it was possible to install special rail switches and monorail hoists (Fig. 3), thus mechanizing the whole flow of materials.

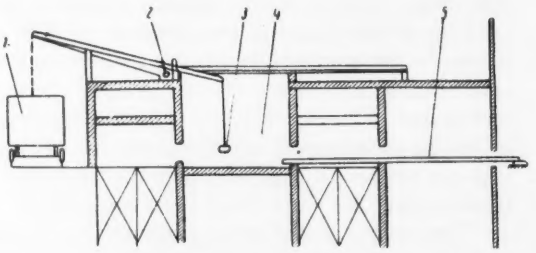

Fig. 3. Monorail hoist employed at the Dzerzhinskii Works: 1) Dumpcar for brick rubble; 2) hoist ; 3) bucket; 4) furnace cell; 5) belt conveyor.

The method of pulling down the separating walls of the regenerative soaking pits employed at the Kuznets Metallurgical Combine should be mentioned. To speed up the demolition work, the ends of the separated walls are cut in at the places where they adjoin the longitudinal walls of the cell, and the whole block (the separating wall) is removed with the hook of the soaking pit crane.

REPAIRS TO CONTINUOUS FURNACES

During the repairs to continuous furnaces all the materials are usually transported by belt conveyors.

The skidding tubes are best changed by the method of the Zakavkaz Metallurgical Works. Separate parts of new skids are welded outside the furnace. The tube

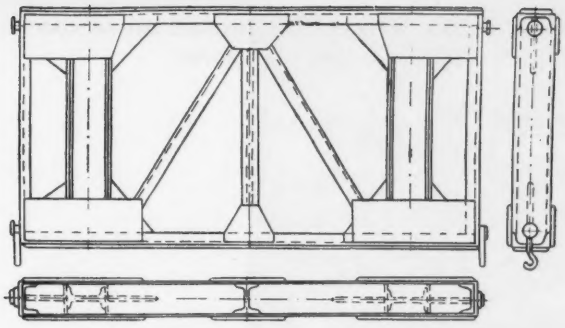

Fig. 4. Frame for pushing the blooms from the furnace.

is brought to the charging door by means of the bridge crane hook. The tube is moved as far as the movement of the crane carriage or the length of the ropes allow and then it is pushed manually on rolls placed on the old skidding tubes. Therefore, the old skidding tubes are not removed until the new ones are installed. In this way, repairs of the continuous furnace can be carried out much faster. During the dismantling, the skidding tubes are cut into pieces by a gas or oil flame.

To remove blooms from the continuous furnaces before the repairs, the Zakavkaz Metallurgical Works employs special frames (Fig. 4) which are charged into the furnace by means of pushers. The frames are joined together so that they can be removed from the furnace after the discharge of the blooms. The number of frames for the discharge of the blooms depends on the length of the working bottom of the continuous furnace. After the blooms are discharged, the frames are pulled out of the furnace by means of the same pusher. They are then separated and transferred to storage.

The adoption of the above methods of repairs to heating facilities with the use of mechanized equipment will help metallurgical works in carrying out planned and systematic repairs. To spread the use of advanced techniques to all works it would be very desirable to arrange interworks courses for the exchange of experience on the repairs to heating facilities.

DUST-CATCHING DEVICE

I.Ya. Yasik

Senior Design Engineer of the Rail-Rolling Plant of the Kuznetsk Metallurgical Combine

In rolling shaped profiles (rails and sleepers), much metal dust is formed. Several designs of water sprays have been tried out in the rolling mills for settling the dust.

Water, atomized by compressed air, is forced through nozzles (Fig. 1), and settles the smallest particles of the dusty atmosphere.

For the most effective dust catching, as much water as possible is necessary, so that all the dust particles are wetted to the maximum extent, and are settled by the water and not by the compressed air. The device was tried out on stand 750 of the rail-rolling mill (Fig. 2). The use of water for removing the dust from the passes of stand 750 gave good results. It has been possible in this way to retain and settle about 70–80% of the dust. Fig. 2 shows that the device is simple in construction. Mains water is fed by a pipeline to a special frame, provided with a large number of orifices 3 mm in diam. at an angle of about 20° to the plane of the bar leaving the pass. A pump is provided in addition

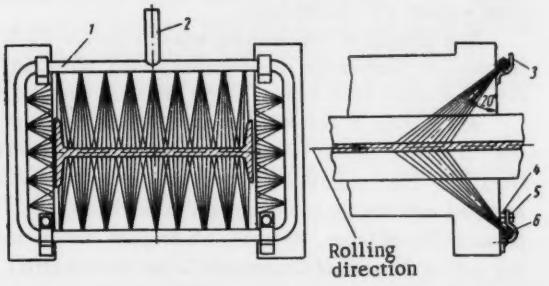

Fig. 1. Diagram showing working principle of atomizers: 1) Frame; 2) pipe; 3) bracket; 4) nut; 5) bolt; 6)clip.

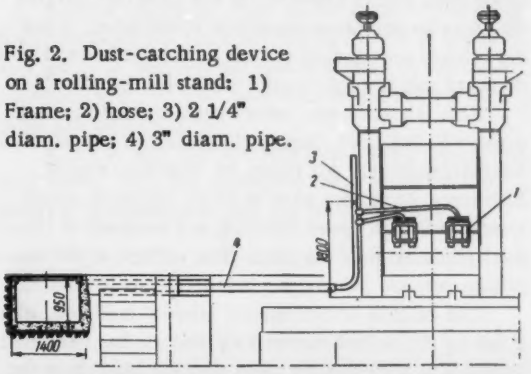

Fig. 2. Dust-catching device on a rolling-mill stand: 1) Frame; 2) hose; 3) 2 1/4" diam. pipe; 4) 3" diam. pipe.

for maintaining constant pressure and increasing the supply of water in the pipeline.

This method is most effective and rational compared with those previously employed.

AT THE BAKU TUBE-ROLLING MILL

R.A. Faibisovich

Construction of the Baku Tube-Rolling Mill began in the difficult years of the Second World War, and the main functions of the plant were put into operation soon after the war's end.

Not long ago Sumgait steel was shipped to Rustavi where it was rolled into circular billets and shipped back. Thousands of railway cars were occupied in such two-way transportation and cost the state considerable money. Installation of a rolling mill made it possible to produce billets at the plant. At present all conditions have been met for a smooth and rhythmic operation of the plant and for increasing the output of tubes. The rolling mill operators mastered the new technological process in a short time. Already in 1958 the production of steel and tubes increased considerably in comparison with 1957.

The plant supplies its products to numerous enterprises here and abroad. Having filled an order for export, the collective of the plant shipped a large consignment of tubes to Bhilai, India, for the construction of a large metallurgical plant. Hundreds of tons of tubes have been sent to the People's Republic of Albania, the German Democratic Republic, the Chinese People's Republic, the Rumanian People's Republic and other countries of the world.

The struggle for technical progress and the endeavor to fulfill successfully the planned tasks favor increasing the qualification of metallurgists. Socialist competition broadens more and more between steel workers, rolling mill operators, and tube-rolling-mill operators.

"We can smelt at a greater rate," says the eminent speed steel worker Sokolov, "if the time for charging the furnaces and other operations is shortened. I and my brigade are pledged to give to the Motherland several thousand tons of high-quality steel over the plan."

This initiative was ardently maintained by steel workers V. Boldyrev, deputy of the Supreme Council of Azerbaidzhan SSR, Ali Pasha, M. Ierafilov, and M. Zeinalov. At present each of them has to his credit almost 50% high-speed smelting and hundreds of tons of steel produced over the plan. The workers at the tube-rolling shop are not lagging behind the steel workers.

The brigade of Communist labor at mill 140, which is led by the skilled master Tagi Kuliev, has delivered hundreds of high-quality tubes over the plan since the beginning of the year, and is one of the first in the shop

Fig. 1. Steel worker of the open-hearth mill, deputy of the Supreme Council of Azerbaidzhan SSR, V.I. Boldyrev.

to fulfill the semiannual plan ahead of schedule. Recently, after carefully studying the possibilities, the brigade decided to work without mechanics and electricians; their responsibilities will be fulfilled by the rolling-press workers and the operators themselves, who will also participate in repairing the tube-rolling mill. This undertaking was approved by the party bureau at the shop, and the engineering-technical workers and shop workers will render help.

The brigade of tube-rolling workers headed by A. Gromov, N. Grebenyuk, V. Miroshnichenko, and many others attained high indices in the socialist competition.

All workers of the plant are well acquainted with the celebrated master rolling-mill operator Gyul'mamed Gasanov. The youthful collective headed by G. Gasanov attained outstanding achievements in increasing the productivity of labor. Recently his brigade was awarded the rank of Brigade of Communist Labor for industrial valor. Every month the rolling-mill operators deliver on the average, 20 tons of billets over the plan instead of ten tons as was pledged. They achieved great successes in the struggle for utilizing hidden reserves, for

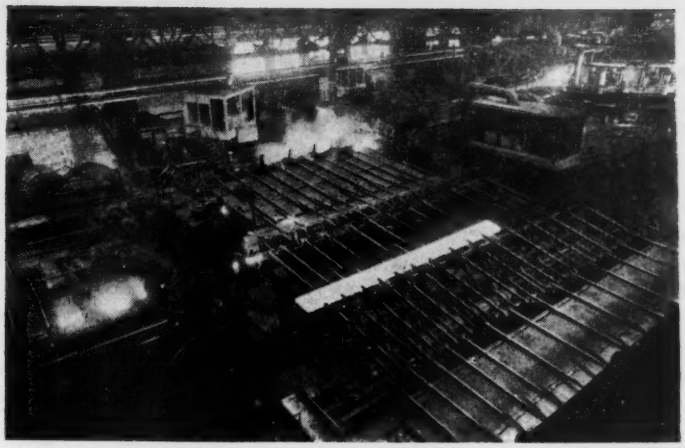

Fig. 2. Hot rolling a tube on mill 250.

improving the quality of the product, and they considerably reduced breakage. The young workers demonstrated great initiative in bringing about cleanliness and order in the shop and plant area. They serve as an example for study and practice.

Great perspectives are opened before the Baku Tube-Rolling Plant in the new Seven-Year Plan. The output of gross products is increased 90%, steel 70%, rolled-goods 90%, and the production of tubes is increased by 120%. Such an increase will be achieved by utilizing new power networks and operational improvements in the mill units, by introducing the suggestions of innovators and inventors, and by carrying out numerous technical and organizational measures.

A demonstration of this is the fact that during 1957 the scientific-technical council of the plant recieved about 400 innovators' suggestions (150 more than the previous year) which saved 2 million rubles, Names well known at the plant are those of: innovator-electrician Comrade Bakhtin, who introduced in the rolling shop 8 suggestions with an annual savings of 96 thousand rubles, and the brigade leader of mechanics for the tube-rolling shop, Comrade Grigoryan, who has to his credit 11 suggestions which were introduced into practice. Great achievements in the field of automation of individual sections of the plant were attained by the group of automation workers in the tube-rolling mill headed by electrician Kremlin. Their last project, the conversion of the charging machine of the annular furnace mill 250 into an automatic system, made it possible to improve the technology of charging and to decrease the staff of operators.

In 1959 new construction began on mill 250-2, a new open-hearth furnace will be operating, and in 1960 the reconstruction of the open-hearth mill will be totally completed.

These operations will entail reconstructing mills 850 and 700. In the rolling shop additional soaking pits will be constructed, the roller tables will be replaced at the shears and a scrap-pusher installed.

Scarfing of the blooms will be mechanized and it is planned to install additional lift cranes. A second soaking pit is planned for the pipe mill and the ascent of the blooms from the storage area to the furnaces will be mechanized.

In 1962 a shop for upsetting the ends of tubes will begin operation. The petroleum industry of the Republic will receive steel drill tubes, the demand for which is great.

One of the greatest measures for increasing the productivity of tube-rolling mills is the introduction of a permanent straightening device which was first tested in the Soviet Union at the Baku Tube-Rolling Mill. The permanent straightener, manufactured of high-carbon steel, showed excellent results having withstood about 2000 passes. As a result of using it the mill will be completely automatic.

Another important event which ensures increasing the productivity of the tube-rolling mills and the shop as a whole is the proposed specialization of the mills, for which mill 250 will be converted for rolling oil pipes and pump tubing.

Mill 140 will manufacture small pump tubing and drill tubes. This will guarantee tubes for the oil industry of Azerbaidzhan and the North Caucasus.

INVESTIGATIONS IN THE FIELD OF THE DIRECT REDUCTION OF IRON FROM ORE

The increase in the cost of scrap and the high capital expenditure for blast-furnace production have led to a search for new methods of the direct production of iron from ore. We have already discussed in our journal* a number of methods for the direct reduction of iron. The following is a description of some new processes which have been proposed and employed abroad† .

Reduction on a belt machine. In this machine (Fig. 1), normal metallurgical equipment is used. The fine ore is mixed with crushed noncoking coal and limestone (the coal and limestone form about 39% of the total charge), and after the addition of a small quantity of binder, the mixture is fed to a plate granulator. The raw pellets are fed to a normal sintering belt. On its passage over the suction chambers of the sintering machine, the charge dries and is ignited, and the oxides of iron react with the coal and are partly reduced to metallic iron (55 - 60%). From the sintering belt, the semi-product passes to an electric arc furnace where iron for steel manufacture is made from it. The temperature of the semi-product is 850°C.

This method has the following advantages:

1) Possibility of using low-grade noncoking coals;

2) Consumption of electricity for melting the iron is 1000–1100 kw-hr/t, which is much less than that for melting pig iron in the electric blast furnaces;

3) Throughput of the complete plant‡ is 400–450 t per day (furnace capacity is 24000 kw). Capital expenditure per ton of iron is 40–50% less than in blast-furnace production.

Currently, the process of belt-machine reduction is in the development stage. The expediency of its use cannot be assessed until industrial trials have been made.

Reduction in a gas current. Three methods are also being developed for the direct reduction of iron ores in a current of reducing gas. They all require carefully prepared, pure ore concentrate.

H-Iron process. The iron ore is reduced in special plants by a current of hydrogen, produced from natural gas or oil, the hydrogen being at a pressure of about 28 atm and the temperature in the reduction chamber being 485°C. The unreacted hydrogen is passed to a drying tower, where it is separated from steam and is then recycled. The reaction produces iron powder, which is subsequently briquetted in a press.

Esso-Little process. The iron ore is reduced by natural gas heated to 325–485°C and mixed with air preheated to 800–850°C . The gas partly burns, and in the ore-reduction zone, which is also preheated, the temperature attains 800–850°C. The unreacted gas is not recycled as in the H-Iron process, but is used as fuel for heating the gas and air used in the process and also for power-generating purposes.

Nu-Iron process. The iron ore concentrate is heated in two steps to 900°C and is reacted with a mixture of CO and H_2 produced by reforming natural gas with steam. The temperature of the mixture is 700°C. After the reduction, the gas is regenerated and recycled.

All three processes described require a concentrate with a content of 66–69% Fe. The economic effect of their application depends on the price of the concentrate and oil or natural gas.

Reduction in rotary kilns. The American firms Republic Steel Corporation and the National Lead Co. have developed a method of reducing iron ore in a rotary kiln at 1000 - 1100°C (Fig. 2). The reducing agent is low-grade solid fuel (coke breeze, anthracite dust and bituminous coals may be used with success). The unreacted reducing agent is regenerated. The kiln is fired by gas or oil. This process has been given the name R-N process after the initial letters of the names of the firms developing it. A plant in Alabama is operating the process with a daily throughput of 175 t of ore containing 34–35% Fe. After reduction, the ore is subjected to magnetic concentration, as the result of which the product is obtained in two forms: a rich product (above 90% Fe) and a poor product (80–82% Fe). The quantity of rich product is 70% of the total output; it is used in open-hearth and electric furnaces. The second form of product is used as rich raw material for blast furnaces. Limestone and dolomite are added to the charge for reducing the sulfur content. The low temperature of the process does not create conditions for the reduction of phosphorus in the metal, and the product has therefore a low phosphorus and sulfur content.

The R-N process has been used for the reduction of rich ores (68–69% Fe). Several thousands of tons of final product, containing 95–96% Fe, have been melted in electric furnaces.

The carbon consumption in the R-N process is about 0.45 t per ton of final product. The heat consumption for heating the ore is 1.5–2.25 million kcals per ton of product and depends upon the degree of preparation

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*Metallurg, No. 10 (1957) [See C.B. translation].

†Journal of Metals No. 12 (1958); Iron and Coal Trades Review No. 4705 (1958).

‡Here and later, the "short" ton (0.97 t) is used.

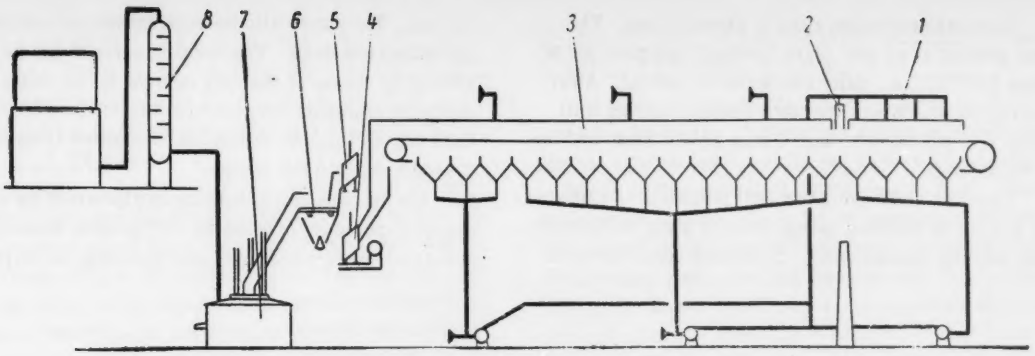

Fig. 1. Diagram of reduction process on a belt machine: 1) Drying section; 2) igniter; 3) reducing section; 4) scales; 5) skip; 6) screen; 7) electric furnace; 8) gas scrubber.

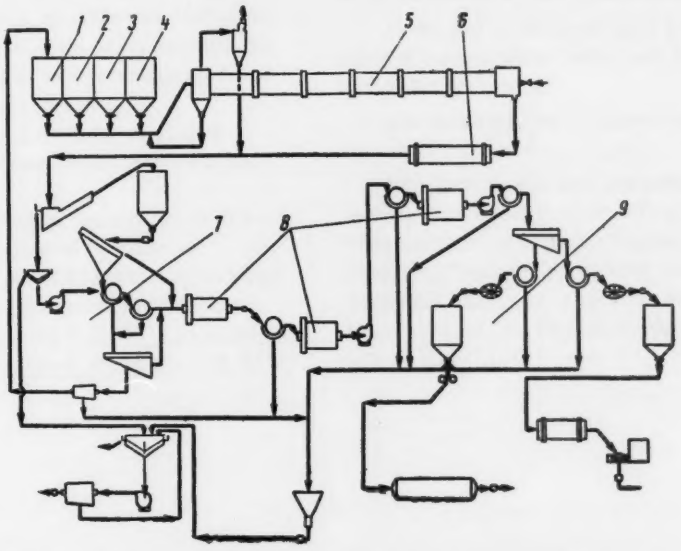

Fig. 2. Diagram of the R-N process: 1) Recycle coke bin; 2) fresh coke bin; 3) ore bin; 4) limestone bin; 5) rotary kiln; 6) cooler; 7) recycle coke group; 8) ball mills; 9) magnetic separation group.

of the ore. The capital expenditure on the construction of the plant is 55–60% that on the construction of a blast-furnace plant of equivalent capacity.

"Hojalata y Lamina" process. The lump ore, sinter or pellets are charged into a reactor and after preliminary heating are covered with a steel cover. Hot reformed natural gas (temperature about 1100°C) is drawn through the ore bed, the ore being reduced to metallic iron. When the ore has been reduced to the required degree, the reactor is discharged and the sponge produced is fed hot to electric furnaces for melting.

This process is being worked at a plant in Monterrey (Mexico), consisting of 5 reactors, each having a capacity of 15 t of ore. The size of the ore lumps charged into the reactor is 6 - 35 mm, iron content of the ore is 66%. The reaction time is 4 hours, and charging and discharging take 30 min. Natural gas, reformed by steam, is used for the reduction, the gas consumption being 700 m^3/t of iron. The quality of the steel from the electric furnace is good. The annual capacity of the plant is 175 thousand tons.

ORF process. Fine rich magnetite concentrates on a travelling grate are heated by waste gases to about 980°C, the magnetite being oxidized to hematite and its temperature rising to 1200°C; the ore cakes slightly without the formation of dust and enters a long, reducing unit. A gas reducer heated to 1100°C is drawn through the bed on the travelling grate, the percentage composition of the gas being:

CO	$CO_2 + H_2O$	H_2	N
70	2	26	2

The total reduction time is about 1 hour. The final product is an iron plate having a temperature of about 1100°C, i.e., quite adequate for rolling. After leaving the machine, the plate enters a rolling mill where it is rolled to a sheet 6 mm or less thick and up to 90 cm wide.

The idea underlying the ORF process is to eliminate the process of melting sponge iron by using sufficiently pure iron ore concentrates. The waste gases are used as fuel. They may also be regenerated and recycled in the reduction stage. The carbon content can be controlled by means of the CO_2 content in the waste gases. Alloying elements may be added to them by introducing their oxides (nickel, cobalt) or ferroalloys (chromium, manganese) into the charge.

For the time being, nothing can be said of the economic expediency of using the ORF process, since there is no actual direct-reduction plant operating on this principle.

M.M. Dizhur. The Automation of Technological Processes in Ferrous Metallurgy. Bibliographical Index. Edited by A.B. Chelyustkin. Moscow, Metallurgical Press, 1959, 214 pages.

The bibliographical index on automation in ferrous metallurgy includes the basic literature (books and journal articles which have appeared in the USSR and abroad) from 1946 to November, 1958. For the convenience of the reader the index has sections explaining the general problems of automatics: control measurement instruments and computing technique and the automation of electric drive. The remaining sections are devoted to the automation of various branches of metallurgy: coke-chemical, blast furnaces, steel smelting, open-hearth furnaces, metallic ware, refractory production. The final section of the index includes the literature on the automation of energetic aggregates of factories in the ferrous metals industry.

The book is provided with an index of authors, which makes it easy to find material of interest to the reader.

Every mention of any book or journal article is accompanied by a brief annotation of its material.

This book can be very useful to workers in factories of the ferrous metal industry, inventors and innovators.

N.G. Starobinskaya and N.I. Agafonova. Electrical Steel Smelting Production. Bibliographical Index. Moscow, Metallurgical Press, 1959, 163 pages.

In this bibliographical index, "Electrical Steel Smelting Production," there is a listing of books and journal articles published during 1950-1958 explaining problems of the electrical metallurgy of steel. The first section contains material of a general character. The second section includes material on the construction of electrical steel smelting furnaces and their auxiliary structures. In the third section are materials explaining the technology of smelting in electrical steel smelting furnaces. The last (fourth) section of the book is devoted to the organization and economics of electrical steel smelting production. Every mention of any material is accompanied by a brief annotation of it.

At the end of the book there is an alphabetical index of authors whose works have been listed in the bibliographical index.

The index will be a valuable help to workers in electrical steel smelting plants.

Metallurgists who wish to purchase the bibliographical indexes "Electrical Steel Smelting Production" and "The Automation of Technological Processes in Ferrous Metallurgy" should apply to the Central Scientific-Technical Library of Ferrous Metallurgy at the address: Moscow, G-34, 2-i Obydenskii per, Dom No. 14. Books will be sent C.O.D.

SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosénergoizdat	State Power Engr. Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LÉIIZhT	Leningrad Power Inst. of Railroad Engineering
LÉT	Leningrad Elec. Engr. School
LÉTI	Leningrad Electrotechnical Inst.
LÉTIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MÉP	Ministry of Electrotechnical Industry
MÉS	Ministry of Electrical Power Plants
MÉSÉP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTi	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NIIZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOÉ	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIÉL	Central Scientific Research Elec. Engr. Lab.
TsNIÉL-MÉS	Central Scientific Research Elec. Engr. Lab.-Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIÉSKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZÉI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us - Publisher.